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Final Report

TASK 5 - OPTIMIZATION OF BLAST DESIGN FOR FOR BLASTHOLE MINING

by

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EXECUTIVE SUMMARY

The project entitled "Optimization of Blast Design for Blasthole Mining Operations", was a contract under the Canada/Manitoba Mineral Development Agreement-Part III, Task 5. The consensus between Inco and Government personnel was that research needed to be performed in this area of science in order to produce a working tool that could be used by all underground blast designers. A three year research and development project was initiated beginning in May, 1986. Emphasis was placed on providing a solid literature review, a data collection system, a review of available blasting models, and developing a user-friendly software package.

Many factors potentially influence rational blast design. Rock mass properties (ore and waste), stope geometry, pre-blast stress levels, and directions, explosives properties, methods of initiation, blasthole layouts, and hole deviation all contribute to fragmentation, blasting costs, wall stability, dilution and scaling requirements.

Although considerable efforts have been expended to research these factors in open pit mines, the technology is not immediately applicable underground. The stress regime is markedly different, the holes are longer, and accurate loading is more difficult to achieve. The post-blast condition of the stope is not generally monitored to determine its relation to the actual execution of the blast.

The intent is that the benefits of this study will be numerous. Optimal fragmentation implies minimization of drawpoint delays, dilution, and wall instability. The effects on costs are significant in that drilling and blasting operations can greatly influence workplace costs.

The driving force behind the project was the main objective of developing a rational blast design that incorporated the important properties of the rock to be blasted, the stability of the resultant opening, and the properties of the explosives and methods of use.

SOMMAIRE

Le projet intitulé "L'optimisation du design des sautages pour la méthode d'exploitation minière par longs trous" était contractuel sous la troisième partie de l'entente de développement minéral Canada - Manitoba, Activité 5. Un accord entre Inco et le gouvernement avait pour objectif de promouvoir la recherche dans le domaine de la technologie minière afin de produire un outil de travail qui pourrait être utilisé par tous les concepteurs de sautage en souterrain. Un programme de recherche et de développement d'une durée de 3 ans a été implanté au début de mai 1986. L'emphase a été portée sur une solide recherche bibliographique, un système de récolte de données, une revue des modèles de sautage déjà existant et la développement d'un logiciel facile d'utilisation.

Plusieurs facteurs potentiels influencent le design des sautages. Les propriétés des roches (minerai et stérile), la géometrie des chambres d'abattage, les niveaux et les directions des contraintes dans la roche avant le sautage, les propriétés des explosifs, leurs méthodes d'initiation, la disposition et la déviation des trous contribuent tous à la fragmentation, aux coûts de sautage, à la stabilité des murs, à la dilution et à la necessité d'écailler.

Bien que des efforts considérables ont été investis dans la recherche de ces facteurs pour les opérations à ciel ouvert, la technologie n'est pas directement applicable pour les opérations souterraines. Une différence majeure existe dans les niveau des contraintes. Les trous de forages sont plus longs et la disposition des charges est difficile à réaliser. L'état de l'excavation après le sautage n'est généralement pas relevé pour ainsi être capable de déterminer les relations avec l'exécution de l'abattage.

Les bénéfices de l'étude seront nombreux car la fragmentation optimale implique la minimisation des délais aux points de soutirage, de la dilution et de l'instabilité des murs. Les effets sur les coûts sont significatifs dans les opérations de forage et sautage et peuvent influencer grandement les coûts des lieux de travail.

L'objectif principal du projet était de développer un système de conception des sautages incluant les propriétés importantes de la roche à être sautée, la stabilité de l'ouverture créée, les propriétés des explosifs et leurs méthodes d'utilisation.

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1.0 INTRODUCTION

Optimization of blast designs can be a rigorous exercise that involves numerous calculations using parameters that probably change from workplace to workplace. Furthermore, the results of any particular blast will have large effects on the production rates, mining cost, and dilution of the blasted ore. This project was initiated to provide a rational blast design tool for underground mines and their blasting personnel. Therefore, the primary objective was to develop an user-friendly software package giving a good first estimate of a suitable blast design and capable of meeting day-to-day production requirements.

Perhaps the most important item in consideration of the above was that the methods used in the blast design could be optimized in a logical fashion using locally determined results. Given the preceding statement, the main objective was to develop a user friendly computer program. Thus, the BOULDER^{TM} (Blast Optimization Using Locally Determined Engineering Results) blast design optimization software was produced (Figure 1).

2.0 TECHNICAL DISCUSSION

The philosophy adopted was based on determining the rock mass characteristics, developing a simple design method, and back analysis to calibrate the methods and justify their use. Essentially, successful blasting in an underground mine can be considered a function of blasting and development costs, fragmentation, and damage. The two areas that influence the probability of success are the engineering design and the operations ability to transfer the proposed design into the stoping block.

The first step in determining whether or not a given blast was successful involves data collection. This data will provide for input to the blast design program. Following this, a first estimate of a blast design should be determined, and then modified to suit local constraints in order to help find the optimum results. Finally, calibration of the model with the measured results must be completed. The calibration will ultimately lead to a better understanding of the limitations of the blast design software and thus, yield better and better results each time the designer makes changes.

2.1 Data Collection

The majority of data collection took place in the area of fragmentation results including both the inherent (pre-blast) fragmentation and the muckpile (post-blast) fragmentation. Emphasis was also placed on blast performance monitoring as a measure of timing and borehole deviation. Rock mass properties used in the project were in the most part previously determined and are discussed in (1). Blast geometry data at Thompson Mine is summarized in (2) and represents a variety of blasting styles. Explosive properties used in the model are simply a reproduction of data supplied by the various explosive distributors.

2.1.1 Muckpile Fragmentation

With the primary area of concern being that of a qualitative and quantitative fragmentation prediction model, two data sheets were set up. Firstly, a sheet as shown in figure 2, which would enable the collection of muckpile fragmentation data was produced and included the following:

1) Size ranges and proportion of each size

2) Orientation of joint planes and frequency of each

3) Shape of muck and proportions of each

4) Secondary blasting details

5) Drawpoint mucking times

2.1.2 Inherent Fragmentation

The second data sheet was produced based upon a revised scanline survey whereby the dip and dip direction of a predominant joint set are measured once and then similar joints counted along the scanline. The procedure carries on until major sets and random joints have been accounted for. The data can then be transformed into a prediction of the inherent fragmentation using a negative exponential distribution. A full description of the method is presented in (3). The data collection sheet is as shown in figure 3.

2.1.3 Blast Monitoring

In the area of blast performance monitoring it was decided that an independent group be brought to Thompson Mine under a separate contract with Inco. The field testing was completed in May 1987 by the Julius Kruttschnitt Mineral Research Centre from Australia, and yielded some very interesting results in regards to timing, hole to hole interference and overall blast performance. The results of the detonator firing times are shown in Table 1.

2.1.4 Rock Mass Properties

Rock mass properties used in the model are mainly derived from previous work. Additional tests were completed by C.I.L. to determine some of the dynamic properties of the common ore zone rocks. The summary of results of the rock type tests is shown in Table 2. The current list of rock mass properties used in the model is displayed in Table 3.

2.1.5 Explosive Properties

The explosive properties that can be input to the model are a collection of the most commonly used industrial properties. They are very much dependant on the information supplied by the individual explosive manufacturers. A typical set of parameters used by the model is given in Table 4.

2.2 Design Methods

The most important phase of the project included the development of design methods that produce good first estimates and then enable adjustments to be made based on the actual results. Four areas of development were forecasted, and then a literature review (Appendix A) was completed to find appropriate models to build on. These included blast design analysis, fragmentation analysis, damage analysis and timing/charging analysis.

The blast design model is simply a collection of the important parameters for a given blast that interact with, and influence, each other until a compromise is reached between the design engineers requirements and the production constraints.

2.2.1 Empirical Powder Factor

The process begins with the determination of a rock powder factor that is based on a relationship between an estimate of the rock mass strength (RMS) and the powder factor found by experience to yield favourable results. A graph of this relationship is shown in figure 4. The equation used to represent this experience is given as:

$$\text{Powder Factor}_{(\mathbf{Rock})} = k + A^*(\text{RMS}) + B^*(\text{RMS})^2 + C^*(\text{RMS})^3$$

where, $k = 162\ \text{gm/m}^3$
$A = 12.06$
$B = -0.112$
$C = 0.00035$

The rock mass properties used for the RMS value include:

a) RMR or Q
b) UCS (MPa)

and are used in the equation given by Laubscher (5) as:

$$\mathrm{RMS} = \mathrm{UCS} * \frac{(\mathrm{RMR} - \mathrm{UCS\ Rating})}{80} * 0.8$$

2.2.2 Design Powder Factor

The design phase begins with the rock powder factor calculated above and then adjusts this until all of the blast designers constraints are met. The next step involves using the blasthole diameter, explosive density and the powder factor to determine the pattern size given as spacing * burden. The equation is as follows:

$$\text{Blast Pattern}_{\mathbf{(Rock)}} = \frac{\text{Blasthole Area} * \text{Density}_{\mathbf{(Anfo)}}}{\text{Powder Factor}_{\mathbf{(Rock)}}}$$

where, Blast Pattern$_{\mathbf{(Rock)}}$ (M^2) = Spacing(M) * Burden(M)

This pattern must then be modified to account for the explosive coupling and the explosive type (if Anfo not used). The manner for accomplishing this involves relating the relative energy of the modified explosive properties to that of the Anfo loaded borehole. The equation for this process is described by AECI (6) and is given as:

$$\text{Relative Energy}_{\mathbf{(Anfo)}} = \frac{\mathrm{Mc \times RWS}}{709}$$

where, Mc = charge mass/metre
RWS = relative weight strength of explosive used

The equation has been somewhat modified in the model to include the coupling efficiency of the explosive and gives:

$$\mathrm{Mc} = \text{Blasthole Area} * \text{Density} * \text{Coupling Efficiency}$$

Modification of the pattern based on the rock powder factor into one that takes into account the new explosive properties occurs as follows:

$$\text{Blast Pattern}_{(\text{Expl})} = \left(\frac{RE_{(\text{Expl})}}{RE_{(\text{Anfo})}}\right)^{2/3} * \text{Pattern}_{(\text{Rock})}$$

which is also described in AECI (6).

Borehole deviation is the next parameter which will affect the pattern. If deviation is excessive, the burden that the explosive must overcome may be too large, whereby the pattern size must be decreased. An example of a test stoping block is shown in figure 5. This adjustment is equated as:

$$\text{Blast Pattern}_{(\text{Dev})} = (\text{Pattern}_{(\text{Expl})})^{1/2} - 0.1 - (\text{BHDV}*\text{BHL})$$

where, BHDV = borehole deviation (M/M)
BHL = blasthole length (M)

The degree of fixation encountered by the blast pattern can further reduce the design pattern. Fixation is the constraint put on the blast by the unavailability of a free face (7) and is presented in equation form as:

$$\text{Blast Pattern}_{(\text{Fix})} = \text{Pattern}_{(\text{Dev})} / \text{Fixation Factor}$$

The last constraint on the pattern is the orebody width. The model attempts to fit the derived pattern (specifically the blasthole spacing) across the stoping block. If the iteration process yields less than 5 holes across the orebody width then an additional correction is made to the powder factor to account for the swell factor (8). This process is displayed below.

$$\text{Powder Factor}_{(\text{Design})} = \text{Powder Factor}_{(\text{Rock})} + \text{SW}$$

where, $\text{SW} = \dfrac{\text{Powder Factor}_{(\text{Rock})} * 0.9}{\text{Number of holes/row} - 1}$

The pattern is then reduced in size based on the ratio of the rock powder factor and the new powder factor in the equation:

$$\text{Blast Pattern}_{(\text{Swell})} = \text{Blast Pattern}_{(\text{Fix})} * \text{PFA}$$

where, $\text{PFA} = \dfrac{\text{Powder Factor}_{(\text{Rock})}}{\text{Powder Factor}_{(\text{Design})}}$

Once the acceptable pattern is found that meets the specification of the designer, the final powder, spacing, burden and number of holes across the row are output.

2.2.3 Fragmentation Model

The blast design model attempts to answer the question, "Will it break?". The next question to be addressed is "How will it break?". The fragmentation model will take the output from the blast design model and additional rock mass properties to formulate an answer to the queries above.

- **Inherent Fragmentation**

The most important factor identified by the project in relation to the final muckpile fragmentation is that of the inherent fragmentation (natural fractures, etc.) in the rock mass. This portion of the model is fully described in Appendix B and will be only briefly reviewed here.

Essentially, the model takes the three most common joint set spacings and creates a large block containing these joints using a negative exponential distribution. The block is taken apart and sorted to yield a percent passing curve.

- **Kuz-Ram Fragmentation**

Secondly, the Kuz-Ram model as described by Cunningham (9) is used with the various blasting geometry and charging details. This results in a percent passing curve based on a homogeneous or nearly homogeneous block of ground. The next step involves taking the drill pattern and superimposing it over the individual inherent blocks. The probability that the Kuz-Ram model will affect this block is thus determined, and, if high enough, will break the inherent blocks into smaller pieces. The resulting volumes of rock unaffected by this relationship are added to the Kuz-Ram volumes to yield the total volume in a percent passing curve. The key to the model presented above is that both the inherent fragmentation model and the Kuz-Ram model are set up to allow the user to adjust parameters that may be unique to his/her mining block.

2.2.4 Damage Analysis

The next topic that had to be handled was damage. The question that the designer tries to answer is "Will it overbreak?". The contributing factors to overbreak are put forth as the shock and the gas developed from the blast. In this context the shock is a function of borehole pressure and the de-coupling while the gas generated is proportional to the charge mass.

The damage model begins with a calculation of the peak particle velocity based on research done by Holmberg (10) whereby:

$$V = k^{*}L^{\alpha *}\left\{ \int_{0}^{H} \frac{dx}{[R_0^{2} + (R_0 \tan\theta - x)^{2}]^{\beta/2\alpha}} \right\}^{\alpha}$$

This peak particle velocity can then be adjusted for actual borehole pressures based on the equation given in (11) as:

$$BP_{(Charged)} = (C.R.)^{2.4} * BP_{(Ideal)}$$

where, BP = borehole pressure

C.R. = coupling ratio

The user inputs the ideal borehole pressure obtained from the explosive supplier, or as per calculated by various formulas that exist. Also needed are the charge length, borehole length, radius of charge and radius of borehole to determine the value C.R. where:

$$\text{Coupling Ratio C.R.} = C^{1/2} * \frac{r_c}{r_h}$$

$$\text{where, } C = \frac{\text{length of charge}}{\text{length of hole}}$$

Once the borehole pressure for the actual charged hole is known then V may be adjusted whereby the assumption is made that :

$$V_{(Actual)} = V * \frac{BP_{(Charged)}}{BP_{(Anfo)}}$$

where, V = peak particle velocity (mm/sec)

The values of V (actual) are contoured in three ranges as indicated by:

1. definite breakage 2500 mm/sec
2. damage (potential dilution) 1000-2499 mm/sec
3. minimal damage 0-999 mm/sec

Obviously, stronger rock masses are going to suffer less damage at certain ranges and the facility to adjust the spectrum of overbreak is assumed as follows:

$$\text{Contour Damage Limit} = \frac{\text{Powder Factor}_{\text{(Rock)}}}{450} * \text{Known Damage Limit}$$

The intent of the damage model is to give the designer a means of assessing the overbreak and calibrating that with the actual dilution experienced underground.

2.2.5 Blast Performance

The final question the blast designer deals with is "Will the blast perform as planned?". In order to accept a positive answer, an analysis must be completed on the timing and charging details of the blast.

- **Charging Analysis**

In terms of the charging analysis the method chosen was that of powder factor contours. Once the user has input a set of blastholes with the details of geometry, charge lengths and types, a simple inverse distance weighting function contours the results of powder factor for a given block of rock. The designer can then quickly view the results to determine if holes and/or charges should be altered to yield an overall powder factor close to that of the rock powder factor.

- **Timing Analysis**

The second part of the blast performance model contains algorithms developed by Rholl (12). Input for the model consists of the sequence of blasting caps used, nominal firing times, delay variation, and a crowding factor. Calculations begin with the determination of the normal distribution of each delay detonator. Each distribution is then compared to check for overlap in the firing times. The output from the model consists of the probability of success that the cap will fire in the intended sequence, and an overall probability of success.

2.3 Calibration

An important part of the project involved the comparison of the models predicted output to that of actual underground results. Four areas were identified and included:

1) Blast Design
2) Fragmentation
3) Damage
4) Timing & charging

Calibration of these four items was not fully achieved and will be an ongoing process. However, the work that has been verified showed good results.

2.3.1 Blast Design

The most practical way to verify the blast design model was to compare it to actual designs at Thompson Mine. However, this proved to be somewhat difficult in that there were very few instances where a blasthole method of slashing (ie. column loaded) numerous rings was used. The model was tested against the typical 4-1/2 inch and 6 inch Vertical Block Mining(V.B.M.) stopes at Thompson Mine as described in (13). These are stopes that use cratering techniques for blasting, and although initial tests showed promising correlation, more tests must be completed in varying orebody widths. As this report was being generated one or two small diameter blasthole stopes were in production and should lend to making improvements in the calibration of the blast design model significantly.

2.3.2 Fragmentation

- **Inherent**

Two areas of calibration needed to be addressed. These were the inherent and muckpile fragmentation predictions. The main priority for the inherent fragmentation assessment was to produce a scanline survey method which would yield quality data in a fraction of the time for a detailed discontinuity survey. Figure 6 shows a comparison of the detailed method versus the proposed semi-detailed method. Clearly one can see very little difference in the three main joint sets. In terms of checking the inherent size distribution there is no fool proof method in blasthole stopes. Only in a block caving operation where no blastholes are used would a true representation of inherent fragmentation appear in a muckpile. However, there are two items that can be used to give an indication of the exactness of the prediction. The occurrence of joint surfaces on the muck and the average shape of the muck can be calibrated against the information from the scanline survey.

- Muckpile

Firming up the method of muckpile fragmentation prediction involved comparing the drawpoint muckpile size distributions to those predicted by the model. Once again, the data collected was in V.B.M. stoping blocks where a mixture of crater charges and slashing techniques were used. Further work must be done in the calibration of the model in the areas of crater blasting, and also in the recent blasthole stoping blocks.

2.3.3 Damage

Very little work was done in the area of damage assessment in the underground mining horizon. The main reason for this is that there wasn't an effective means of measuring the void left after blasting. Various instruments have been tried by the mine in the past, but yielded inconsistent results and did not endure the rigours of the underground environment. One method that is currently used to estimate the amount of overbreak involves production data. The blast designer can calculate the amount of material that is removed from the stoping block. This method is fully described in (13) and yields results that can be quantified to some extent. However, additional research must be completed to verify the models assessment versus the actual estimated damage.

2.3.4 Timing & Charging

- Timing

Calibration of the timing analysis is essentially left up to the blast designer. There are a variety of delay detonators used in industry. The scatter time inherent to each series varies accordingly. Provisions are made in the model that allow the designer to adjust the accuracy of his or her system, in order to match data that they themselves or others have collected.

- **Charging**

The charging analysis program is basically a contouring package that uses inverse distance (raised to power 'n') weighting to calculate powder factors in a block model. Initially, this value will be a fixed constant until further confirmation can be completed.

3.0 SOFTWARE DESCRIPTION

The required parameters for this project in terms of software capabilities were that the software be suitable for an IBM-PC micro-computer with interactive graphics to assist in blast design functions. The programming language used was Fortran 77 as specified, combined with MetaWindow^{TM} graphics software capable of providing powerful, advanced graphics functions and unsurpassed drawing speed. The software is modular in concept which lends itself to user-friendliness as well as an inherent capability to modify and update the source code. The final version of the Boulder^{TM} software will contain six modules as identified in the main menu as shown in Figure 7. The following is a description of each module and the functions that they perform.

3.1 Data Capture and Maintenance

The primary function of this module is to provide the blast designer a means to input the data particular to his/her minesite. Essentially, the user is prompted by the software to input or update a variety of data libraries which then service the remaining five modules. An important consideration for the above was to give the user a default value and/or the previously used data so that repetition was avoided during consecutive calculations. The menu for this module is displayed in Figure 8.

3.1.1 Property Definition

The parameters for this library include a stope model description, dimensions for the 3-D block model, geometrical mine coordinates, imperial or metric units, and economic factors. This module is basically set up once for each stope or workplace and sets the mathematical constraints for the software.

3.1.2 Rock Mass Properties Database

The data collected by the software in this area is used in the blast design analysis, the fragmentation analysis, and the damage analysis. The input consists of those items related to the rock mass as previously described, and as listed in Table 3.

3.1.3 Explosive Properties Database

A variety of parameters are available for the blast designer to update and personalize their mining situation. The explosive properties are used in the blast design analysis, fragmentation analysis, damage analysis, and the timing/charging analysis. The properties included are listed in Table 4.

3.1.4 Blast Geometry Database

Details of the geometry critical to a particular blast design are input to this library. These include the following:

- a) Spacing
- b) Burden
- c) Height of Block
- d) Length of Block
- e) Blasthole Diameter
- f) Blasthole Accuracy
- g) Blasthole Costs

The data provided is used in the blast design analysis, fragmentation analysis, damage analysis, and the timing/charging analysis.

3.1.5 Timing and Charging Details

The main function of this database collection routine is to provide details for the charging analysis and/or timing analysis. The input consists of explosive and geometrical details for each blasthole or set of blastholes. Also, provisions are made for the inclusion of sequencing information used in setting up the blast design. The information required consists of plug depths, charge weights, stemming lengths and delay detonator numbers.

3.2 Blast Design Analysis

This is the second module in the Boulder™ software package. The primary function of this module is to analyze the data provided by the blast designer in module 1. Figure 9 shows the menu associated with the blast designer analysis. Essentially, there are three areas of concern. These include an empirical blast design, an engineering blast design, and an operating blast design.

3.2.1 Empirical Blast Design

This portion of the blast design analysis simply determines the minimum powder factor necessary to fragment the blast designers chosen rock mass. This rock powder factor may not be enough to provide the degree of fragmentation required by the user, but will give a good first estimate to be used in other routines throughout the software.

3.2.2 Engineering Blast Design

The blast designer is able to apply their own constraints to the blast model in this step. The program will then adjust the powder factor continually until the desired output of a practical blast design is created. From here the designer can move to the fragmentation analysis or charging analysis and back again for additional adjustments. This then becomes the basis for the eventual blast report.

3.2.3 Operating Blast Design

In most cases minor or major adjustments must be made underground to the blasting layouts issued by the engineering department. This next step then provides the blast designer with a means with which to update their model for future analysis and muckpile calibrations. Variances will always be a part of underground mining, and therefore this routine was added to the blast design analysis.

3.3 Fragmentation Analysis

Perhaps the most innovative of the modules is that of the fragmentation analysis. This is the third module incorporated into the Boulder^{TM} software. The menu for this analysis is shown in Figure 10. There are three components for consideration and include inherent rock mass fragmentation, blast design fragmentation, and muckpile fragmentation. Provisions for interactively viewing the fragmentation curves are also included.

3.3.1 Inherent Rock Mass Fragmentation

This routine takes the rock mass properties given earlier and simulates the fragmentation already existing in the rock mass. Calculations are performed that give geometrical output relating to discontinuity spacings and lengths as well as ranges of fragmentation or lack of it. The output provides a means for the blast designer to then assess the effect of his/her blasting constraints on any given rock mass.

3.3.2 Blast Design Fragmentation

The data output by the blast design analysis serves as input to this routine. Once again the effects of the blast geometry are superimposed on the inherent fragmentation and a resulting fragmentation is output for viewing and consideration. The user can interactively update their blast design and then view another fragmentation curve.

3.3.3 Actual Muckpile Fragmentation

This is simply a means provided by the software whereby the user can enter the data on the degree of fragmentation found in the underground drawpoint. The data is then transformed into a curve that can be compared to the inherent and predicted Boulder™ software fragmentation. This will undoubtedly be the most crucial phase in providing a means by which to calibrate the blast design model for any given rock mass and blast design.

3.4 Damage Analysis

A very important consideration in blast designing is the effect of the blast on neighbouring walls and/or drifts. For this reason the fourth module was set up to investigate the damage produced from a given blast. The menu includes two items for analysis including blast design damage and empirical vibration analysis, and are displayed in Figure 11.

3.4.1 Blast Design Damage

This mathematical routine retrieves data from the rocks mass property library, explosives property library, blast geometry library, and the timing and charging database. The above constraints are placed upon the blast and then the results are contoured and output for viewing.

3.4.2 Empirical Vibration Analysis

This function gives a good first estimate of peak particle velocity for a wide variety of distances and charge sizes. The user is prompted for various single blasthole details upon which a calculation is carried out, and then a peak particle velocity is output.

3.5 Timing/Charging Analysis

This module serves two purposes. The first is to give a good estimation of proper sequencing of the delay detonators planned to be used in the blast. The second is to display the results of a charging analysis performed on the details of a blast. The menu for this module is given in Figure 12.

3.5.1 Timing Analysis

The user is prompted for a set of boreholes from the library which are then run through mathematical models described previously. The results are expressed in a graph showing the sequence of delay detonators and the probability of success in terms of the blasting cap firing when it should. The user can then quickly analyze their layout and determine if any changes need to be made.

3.5.2 Charging Analysis

This routine takes information from the timing and charging details database as well as the blast geometry and property definition database. The parameters are introduced to mathematical equations and then the output is contoured to give a visual representation of the results. The results are expressed in terms of energy units that give the user a good impression of heavy or lightly loaded blastholes. The blast designer can then exit to the blasthole database and revise their blasting layout.

3.6 Reporting

The main features of this module are threefold. Firstly, a printed copy of the blast layout report can be obtained. Next, a summary of the costs associated with the blast can be inspected. Finally, a system will be set up whereby the various plot files on the monitor display can be created for third party software which will transmit then to various plotting devices. The menu is displayed in Figure 13.

3.6.1 Blast Layout Report

This portion of the program serves as a method to output some of the input that the blast designer used to determine the blasting parameters needed for a particular workplace. The intent is that this sheet will serve as the blast layout report for the underground staff.

3.6.2 Blasting Pattern Cost

For any given blast design, costs and other economic considerations play a major role. Therefore, this routine was set up so that the user could obtain a good estimate of the cost of the blast pattern chosen. If a particular blast design is more expensive than another, but gives a more reasonable degree of fragmentation, then other considerations become a factor.

3.6.3 Prepare Plot File

The sole purpose of this sub program will be to convert the output files available for monitor viewing to a format suitable for plotting. As the number of plotters has continued to expand in time there is no reasonable means of writing graphics drivers for them all. The intent will be to supply the data in a format whereby a third party plotting package can be used.

4.0 CONCLUSIONS

The concluding remarks are discussed in two separate categories. The first includes a summary of the various phases of the project, while the second gives a brief description of future research and development.

In summarizing, the project had varying quantities of success in each of the three main areas of data collection, model design, and calibration. For the data collection phase a number of different mining horizons in Thompson Mine were surveyed. Even though some of the data has been analyzed and used for calibration there are still further items that may be processed and used in future correlation. Data collection will be an ongoing item in regards to the blast design model as efforts increase to reduce mining costs.

The blast design model was completed to the point that was identified in the objectives of the contract. The model gives a good first estimate of blast design optimization. One issue that slowed progress of the model was that of computer graphics. At the time of signing the contract, the intent was that the software would be suitable for an IBM PC micro-computer. However, since that time the computer industry has made enormous changes to personal computers, and the graphics produced by the model had to keep up. This limited the amount of time needed to make the software as user-friendly as first hoped.

In the area of calibration several items were addressed and some were not. Therefore, work will continue to update the model as new information develops. The calibration phase is very important as the software has been developed in such a manner that it will be better and better as more experience is gained.

Future research and development in the field of blast design optimization is recommended in the following areas:

a) Interactive graphics
b) Autocad compatibility
c) Damage assessment
d) Determination of volume in open blocks
e) Drill hole deviation

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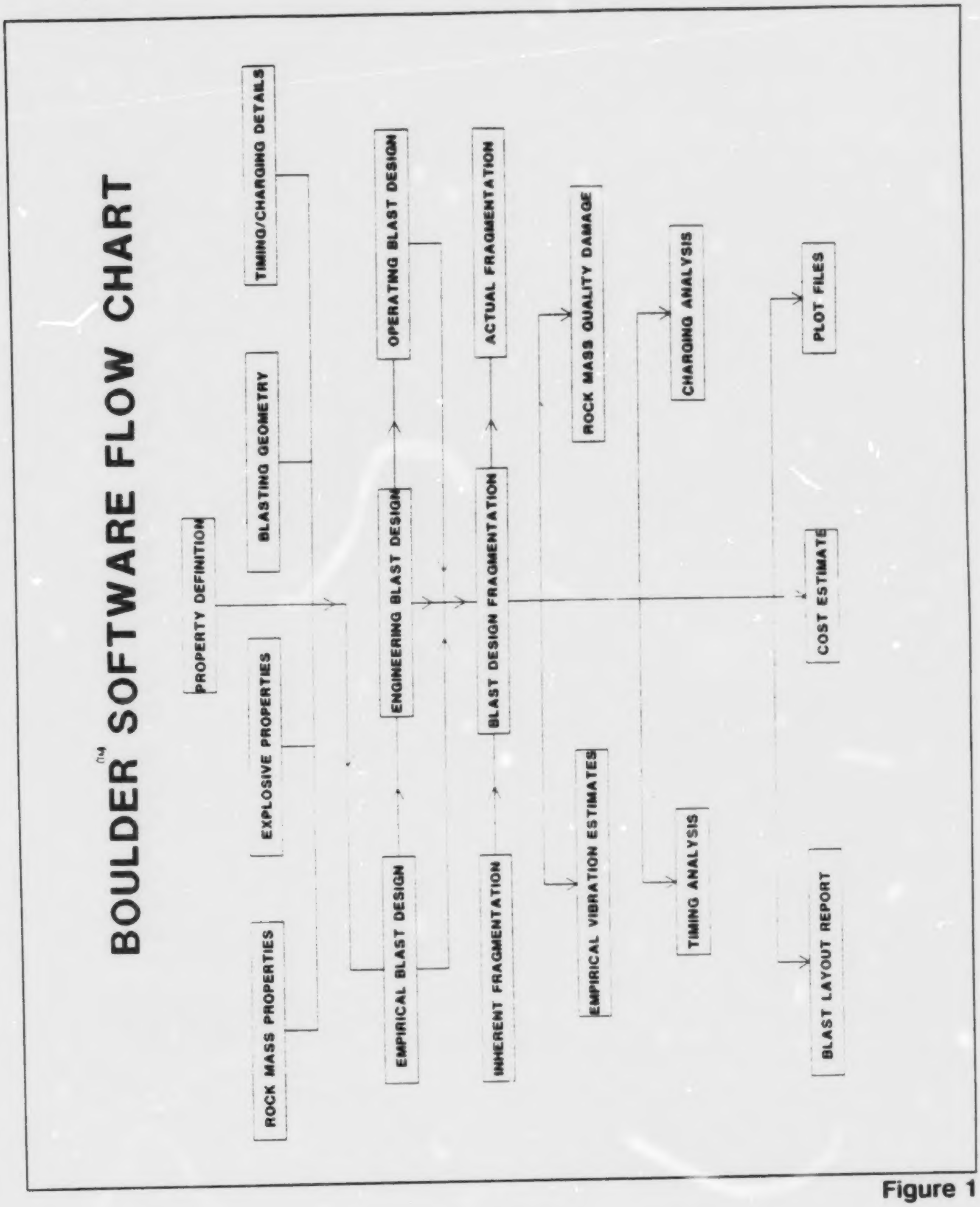
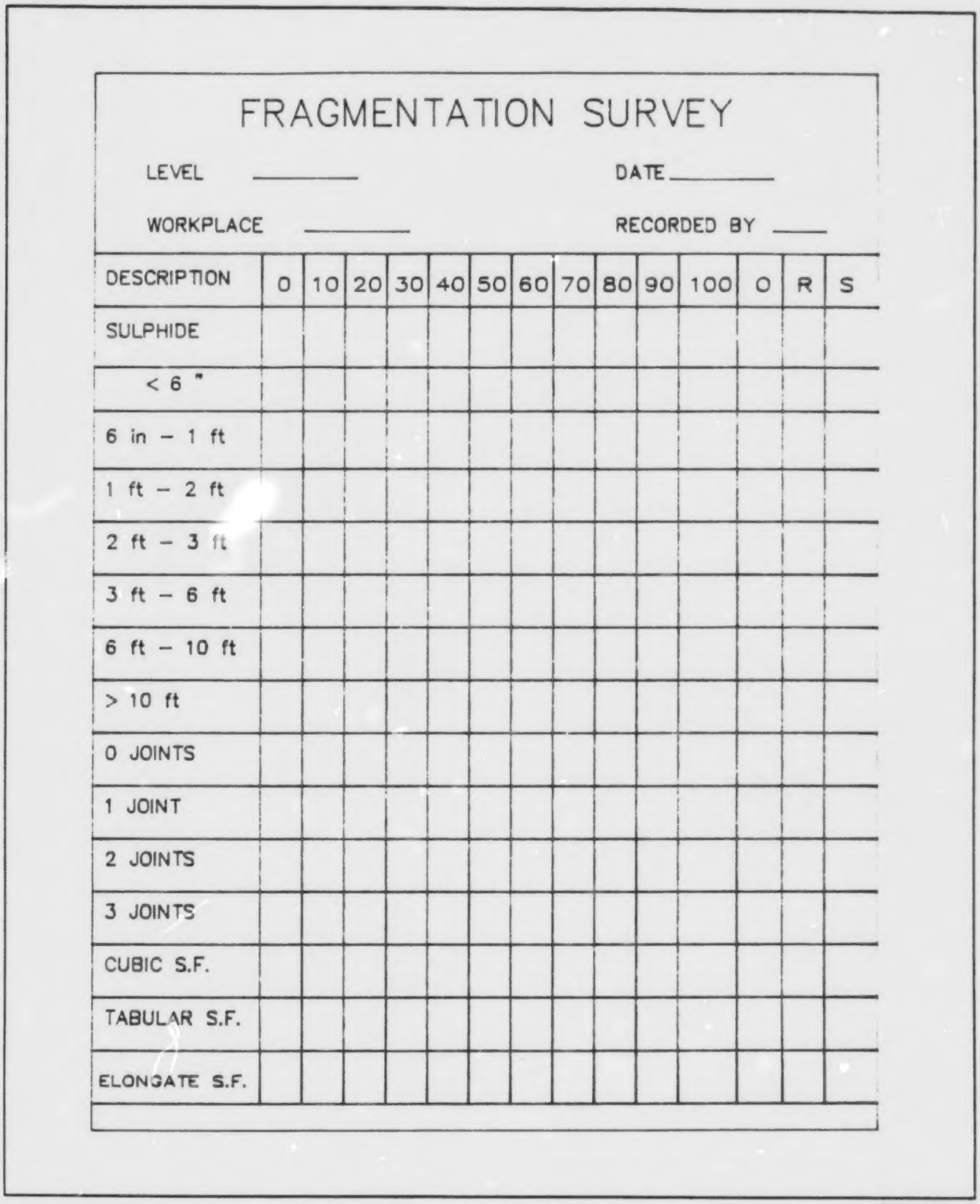

BOULDER(TM) SOFTWARE FLOW CHART
ROCK MASS PROPERTIES
EXPLOSIVE PROPERTIES
PROPERTY DEFINITION
BLASTING GEOMETRY
TIMING/CHARGING DETAILS
EMPIRICAL BLAST DESIGN
ENGINEERING BLAST DESIGN
OPERATING BLAST DESIGN
INHERENT FRAGMENTATION
BLAST DESIGN FRAGMENTATION
ACTUAL FRAGMENTATION
EMPIRICAL VIBRATION ESTIMATES
ROCK MASS QUALITY DAMAGE
TIMING ANALYSIS
CHARGING ANALYSIS
BLAST LAYOUT REPORT
COST ESTIMATE
PLOT FILES

Figure 1



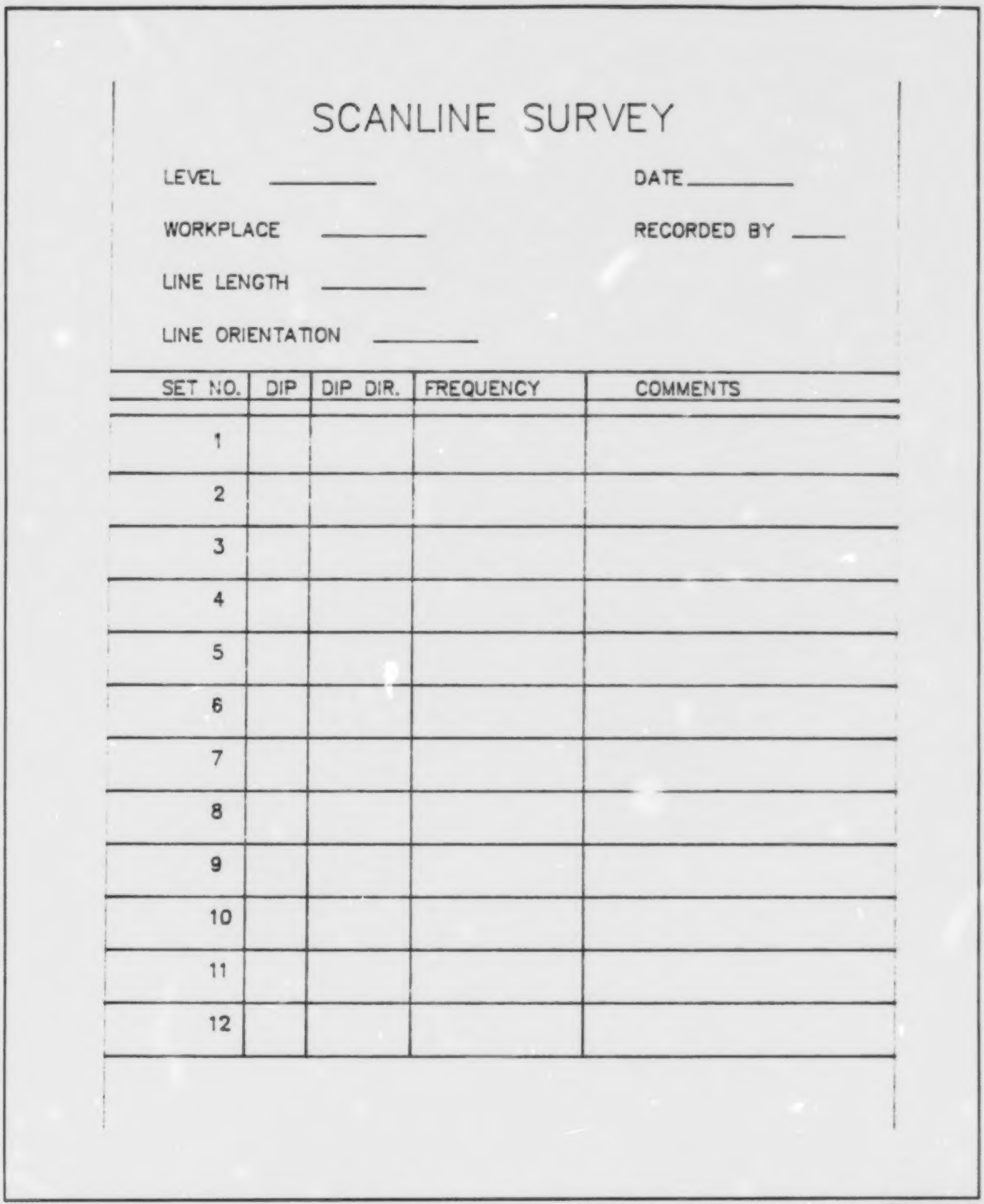
FRAGMENTATION SURVEY

LEVEL ________ DATE ________

WORKPLACE ________ RECORDED BY ____

DESCRIPTION	0	10	20	30	40	50	60	70	80	90	100	O	R	S
SULPHIDE														
< 6 "														
6 in – 1 ft														
1 ft – 2 ft														
2 ft – 3 ft														
3 ft – 6 ft														
6 ft – 10 ft														
> 10 ft														
0 JOINTS														
1 JOINT														
2 JOINTS														
3 JOINTS														
CUBIC S.F.														
TABULAR S.F.														
ELONGATE S.F.														

Figure 2



SCANLINE SURVEY

LEVEL ________ DATE ________

WORKPLACE ________ RECORDED BY ____

LINE LENGTH ________

LINE ORIENTATION ________

SET NO.	DIP	DIP DIR.	FREQUENCY	COMMENTS
1				
2				
3				
4				
5				
6				
7				
8				
9				
10				
11				
12				

Figure 3

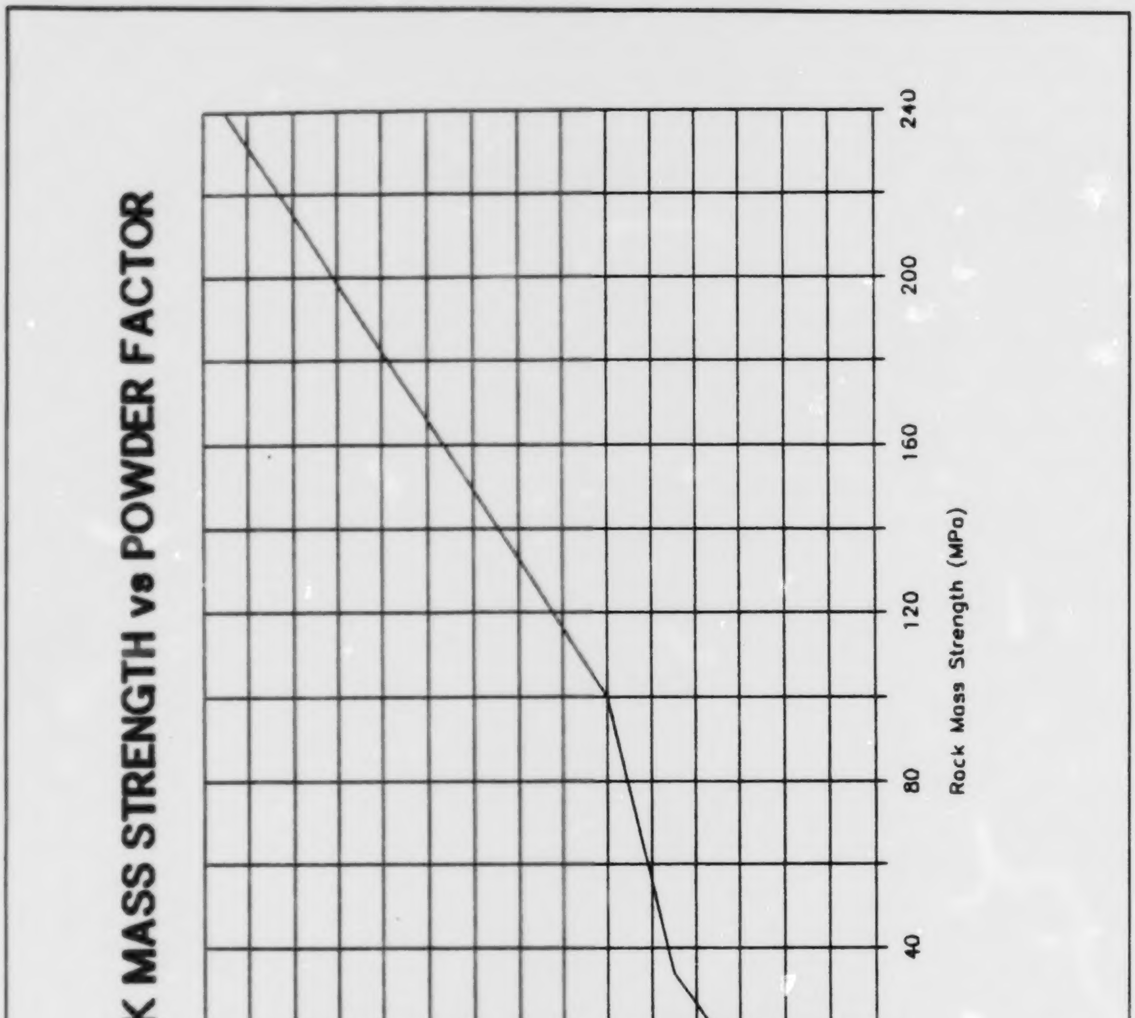

ROCK MASS STRENGTH vs POWDER FACTOR
Rock Powder Factor (g/m^3)
(Thousands)
1.5
1.4
1.3
1.2
1.1
1
0.9
0.8
0.7
0.6
0.5
0.4
0.3
0.2
0.1
0
0
40
80
120
160
200
240
Rock Mass Strength (MPa)

Figure 4

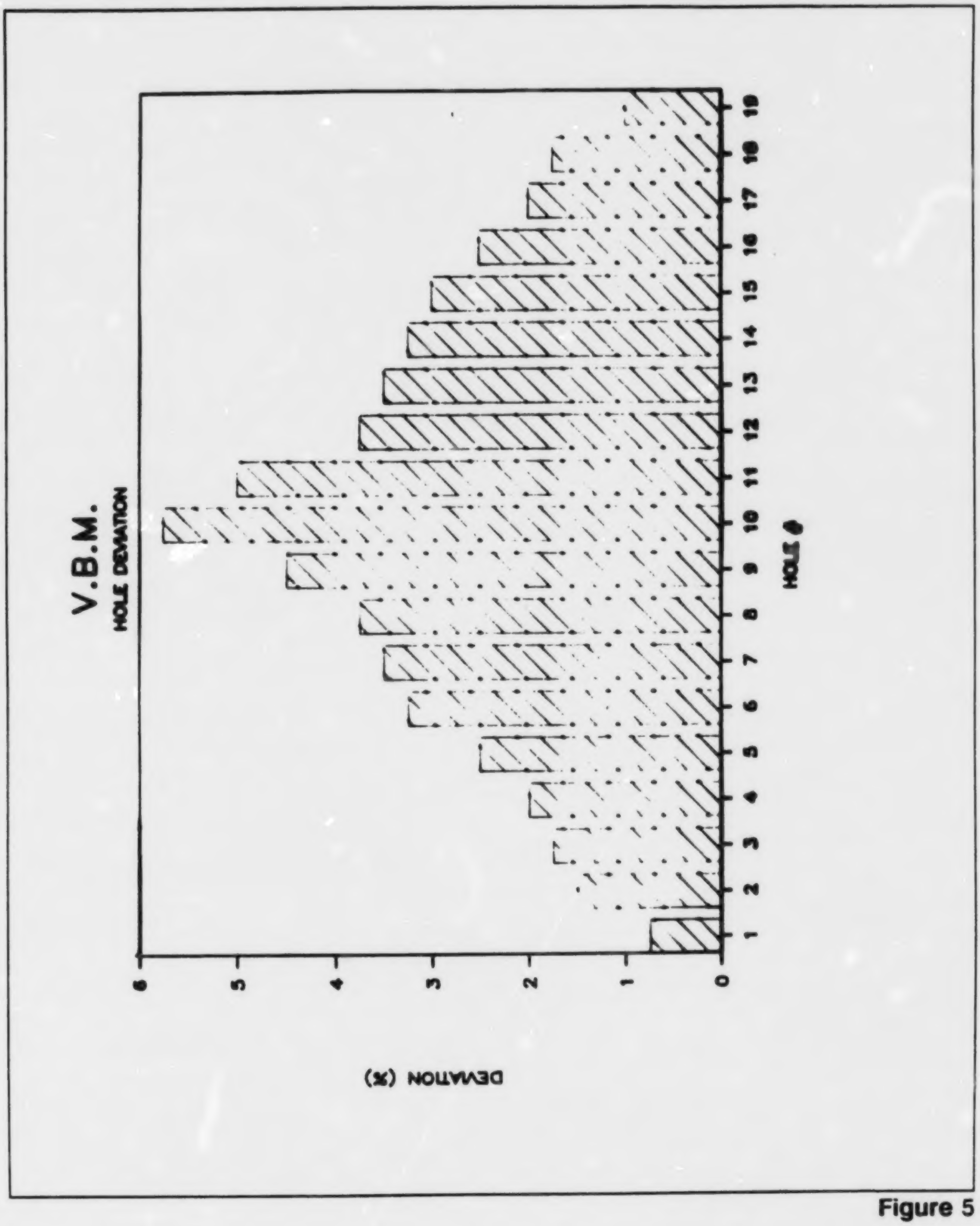

V.B.M.
HOLE DEVIATION
DEVIATION (%)
0
1
2
3
4
5
6
1
2
3
4
5
6
7
8
9
10
11
12
13
14
15
16
17
18
19
HOLE #

Figure 5

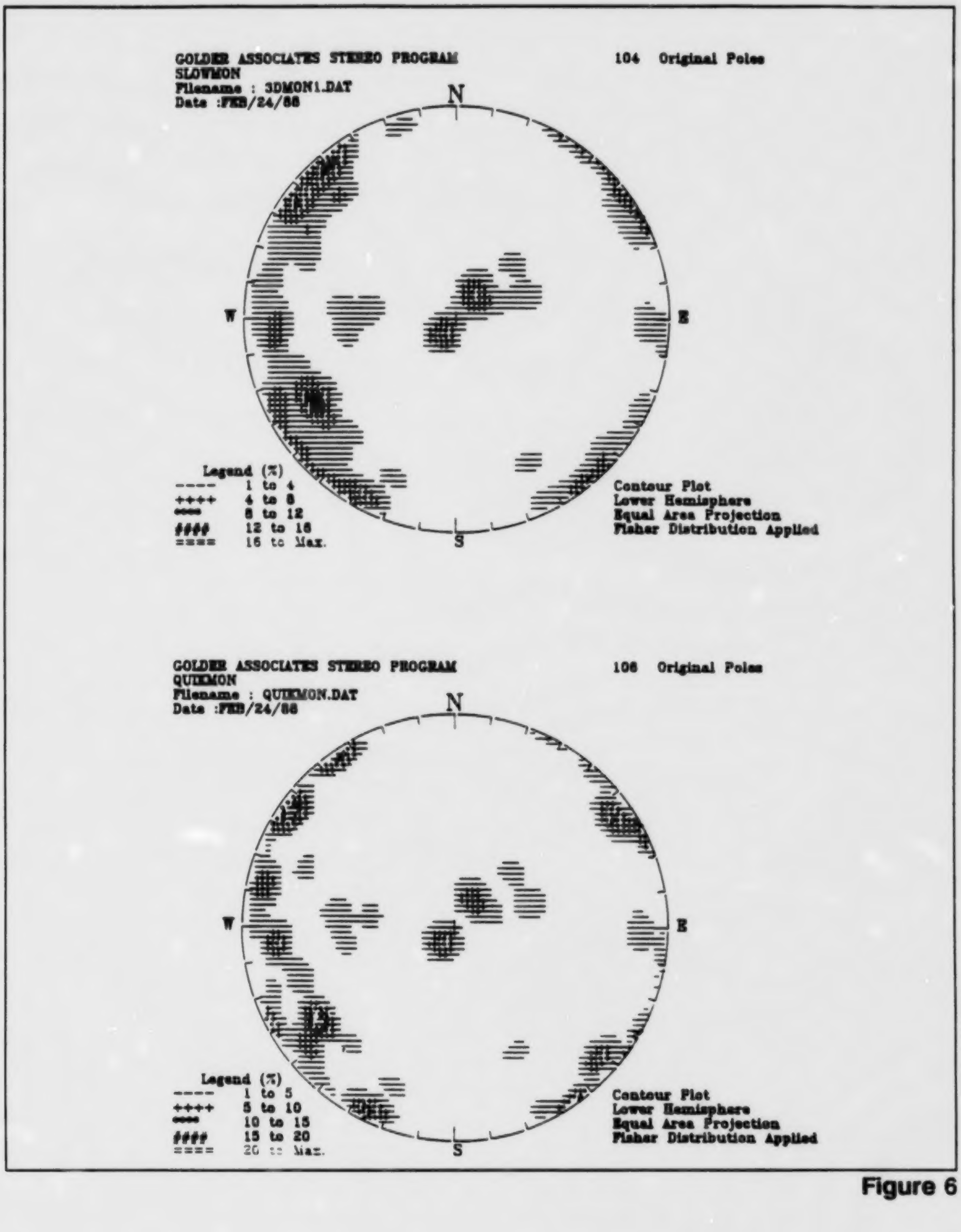
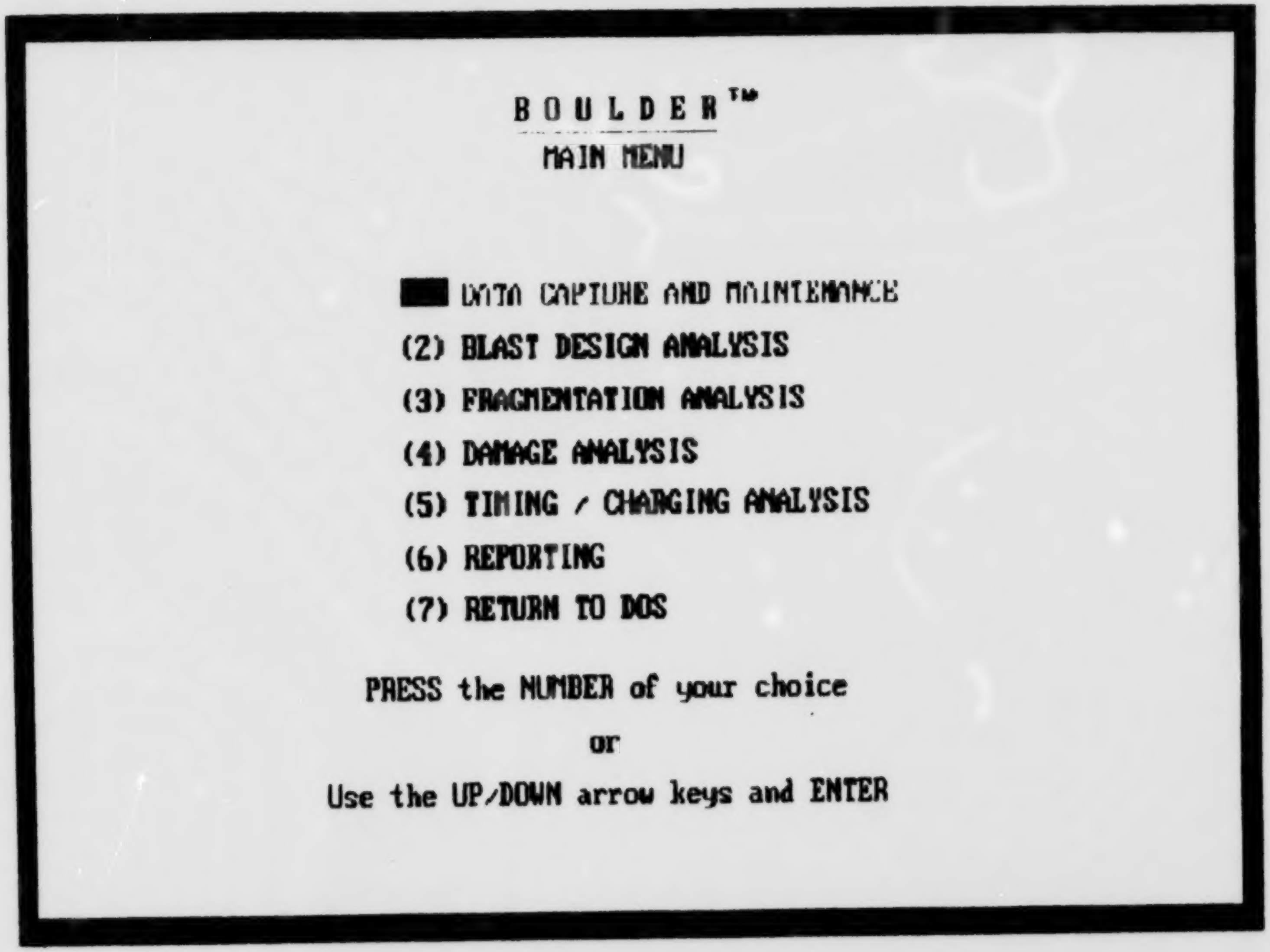

GOLDER ASSOCIATES STEREO PROGRAM
SLOWMON
Filename : 3DMON1.DAT
Date :FEB/24/88
104 Original Poles
N
W
E
S
Legend (%)
---- 1 to 4
++++ 4 to 8
8 to 12
#### 12 to 16
==== 16 to Max.
Contour Plot
Lower Hemisphere
Equal Area Projection
Fisher Distribution Applied
GOLDER ASSOCIATES STEREO PROGRAM
QUIKMON
Filename : QUIKMON.DAT
Date :FEB/24/88
106 Original Poles
N
W
E
S
Legend (%)
---- 1 to 5
++++ 5 to 10
10 to 15
#### 15 to 20
==== 20 to Max.
Contour Plot
Lower Hemisphere
Equal Area Projection
Fisher Distribution Applied

Figure 6

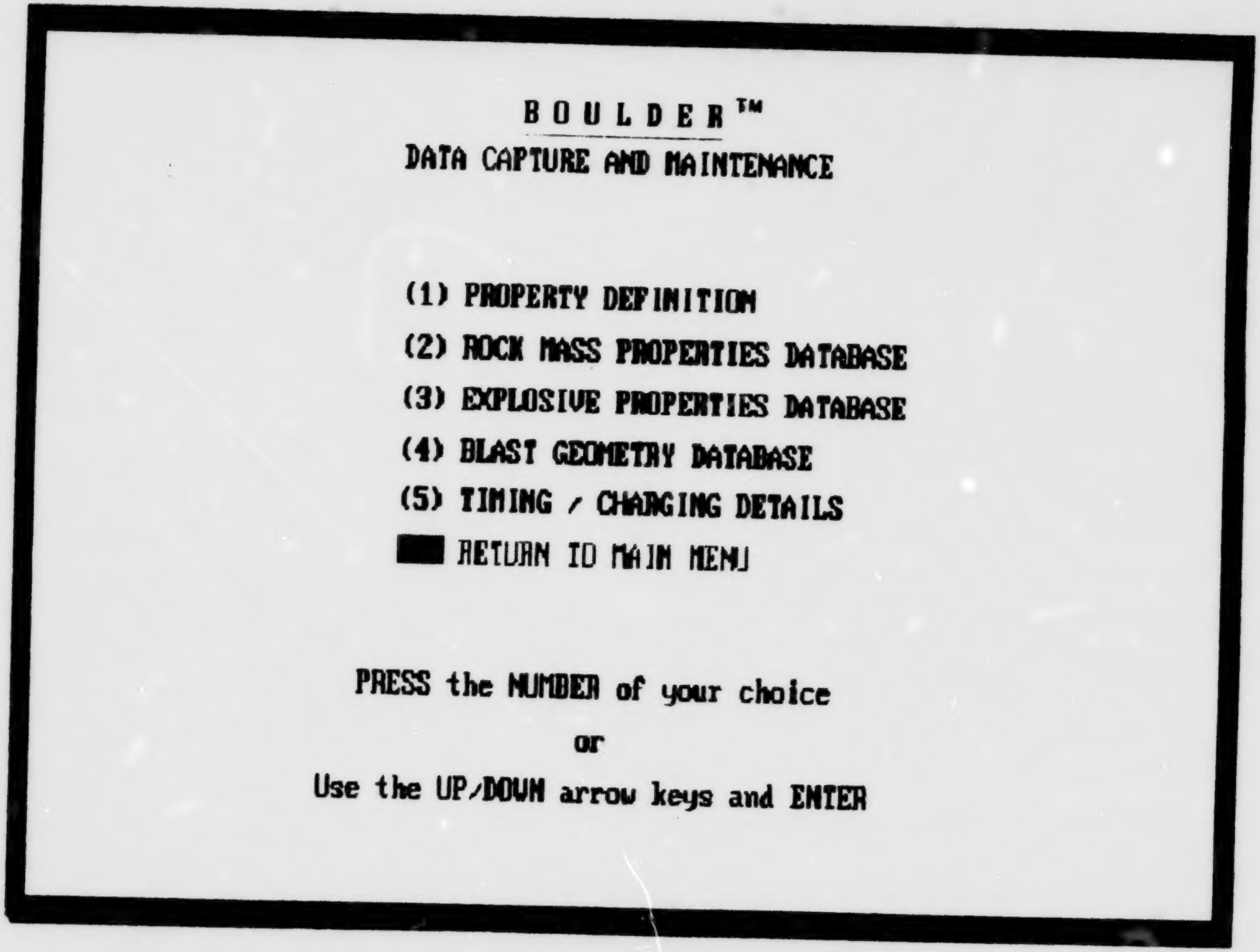


```
          B O U L D E R™
            MAIN MENU

     ■ DATA CAPTURE AND MAINTENANCE
     (2) BLAST DESIGN ANALYSIS
     (3) FRAGMENTATION ANALYSIS
     (4) DAMAGE ANALYSIS
     (5) TIMING / CHARGING ANALYSIS
     (6) REPORTING
     (7) RETURN TO DOS

   PRESS the NUMBER of your choice
                 or
 Use the UP/DOWN arrow keys and ENTER
```

Figure 7



BOULDER™

DATA CAPTURE AND MAINTENANCE

(1) PROPERTY DEFINITION

(2) ROCK MASS PROPERTIES DATABASE

(3) EXPLOSIVE PROPERTIES DATABASE

(4) BLAST GEOMETRY DATABASE

(5) TIMING / CHARGING DETAILS

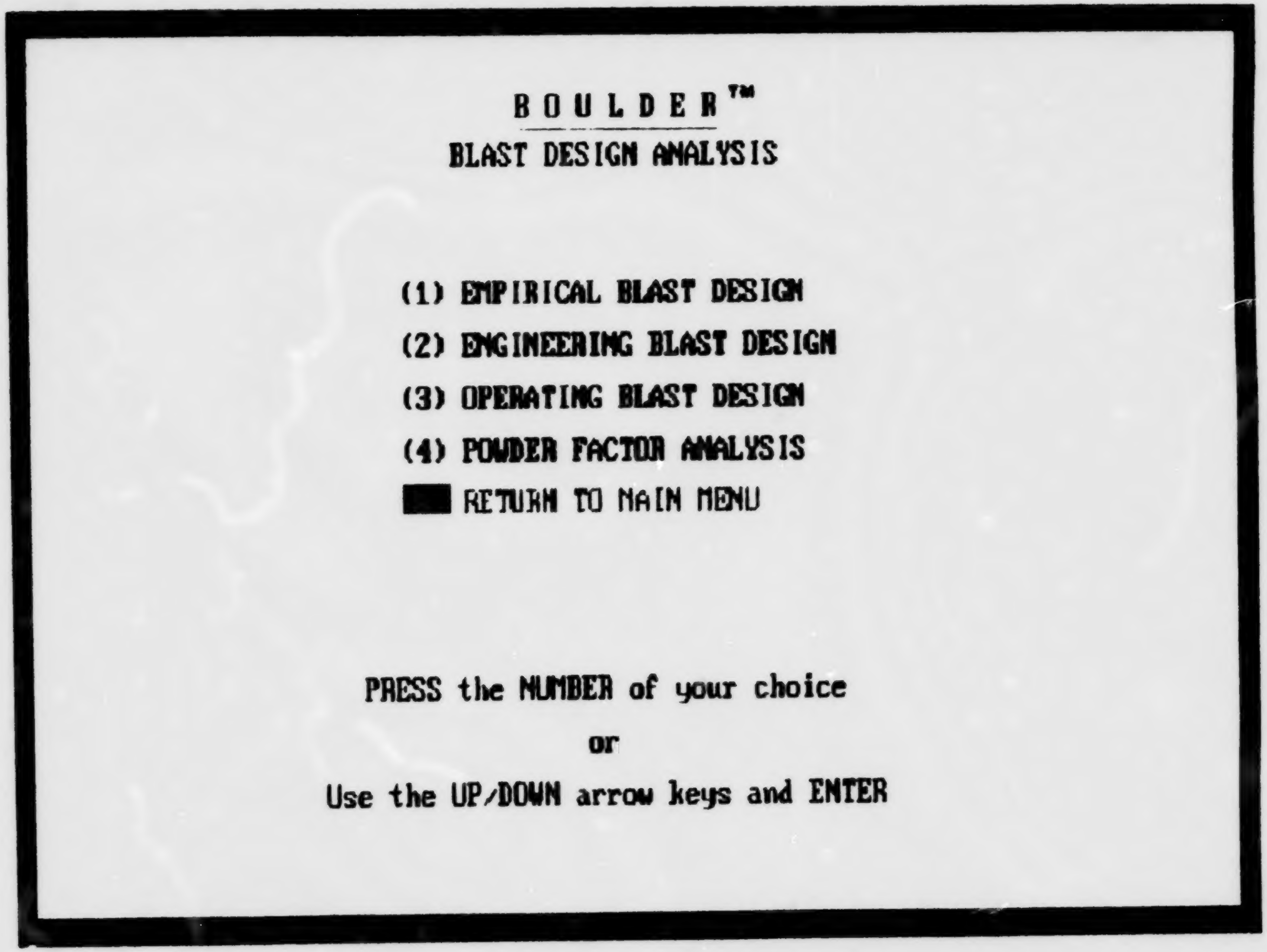
RETURN TO MAIN MENU

PRESS the NUMBER of your choice

or

Use the UP/DOWN arrow keys and ENTER

Figure 8



BOULDER™

BLAST DESIGN ANALYSIS

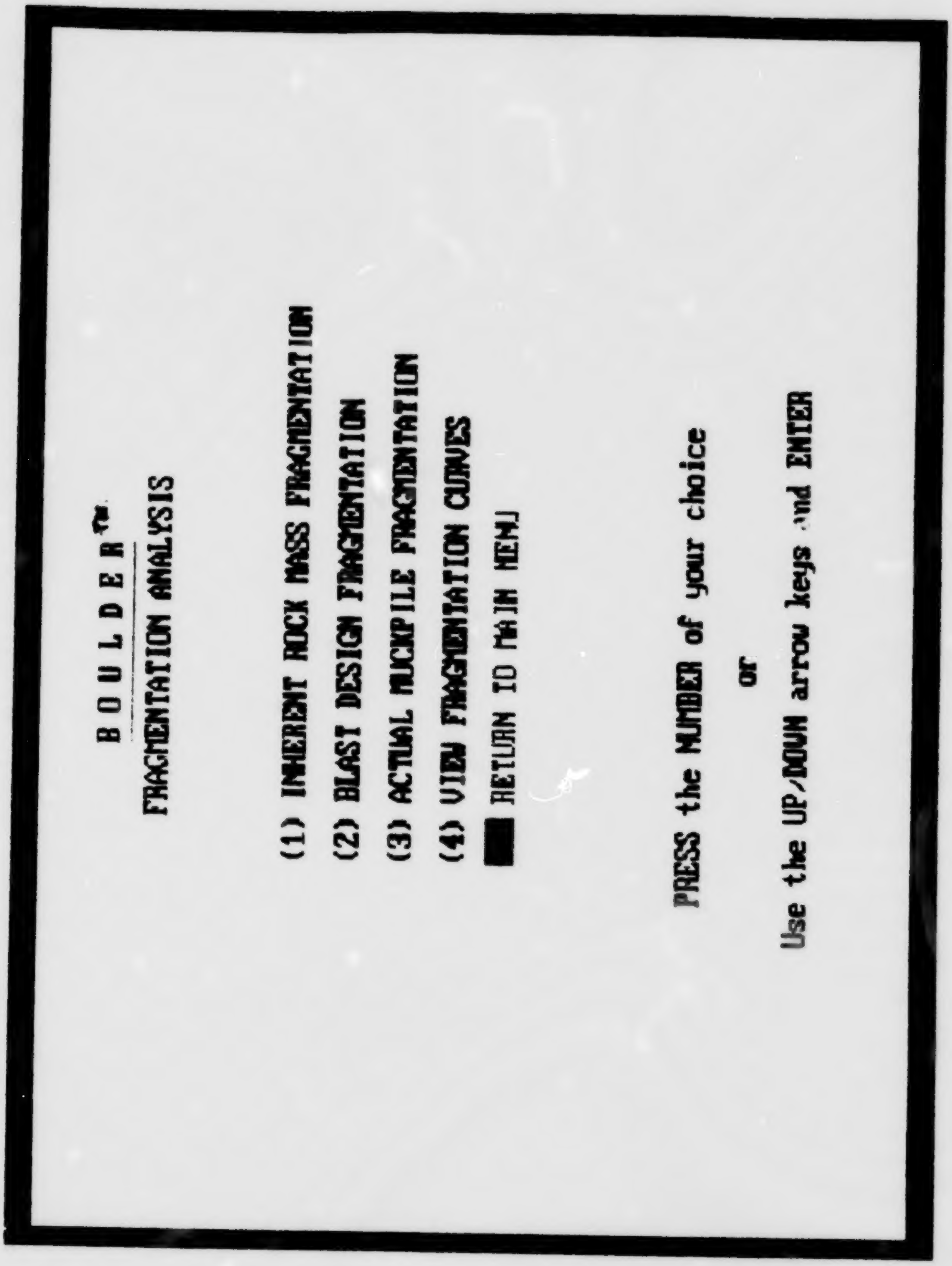
(1) EMPIRICAL BLAST DESIGN
(2) ENGINEERING BLAST DESIGN
(3) OPERATING BLAST DESIGN
(4) POWDER FACTOR ANALYSIS
RETURN TO MAIN MENU

PRESS the NUMBER of your choice

or

Use the UP/DOWN arrow keys and ENTER

Figure 9



BOULDER™

FRAGMENTATION ANALYSIS

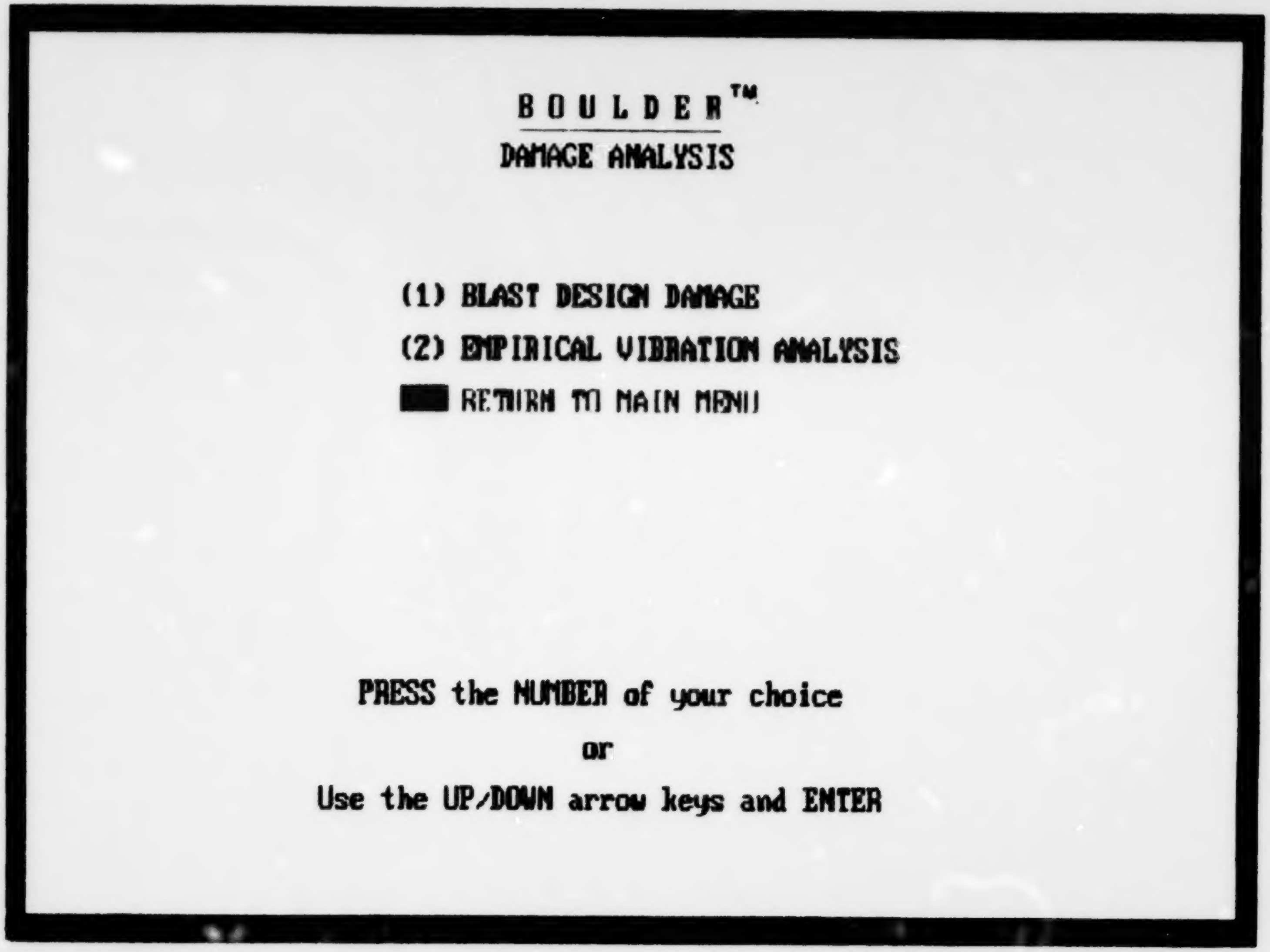
(1) INHERENT ROCK MASS FRAGMENTATION
(2) BLAST DESIGN FRAGMENTATION
(3) ACTUAL MUCKPILE FRAGMENTATION
(4) VIEW FRAGMENTATION CURVES
■ RETURN TO MAIN MENU

PRESS the NUMBER of your choice

or

Use the UP/DOWN arrow keys and ENTER

Figure 10

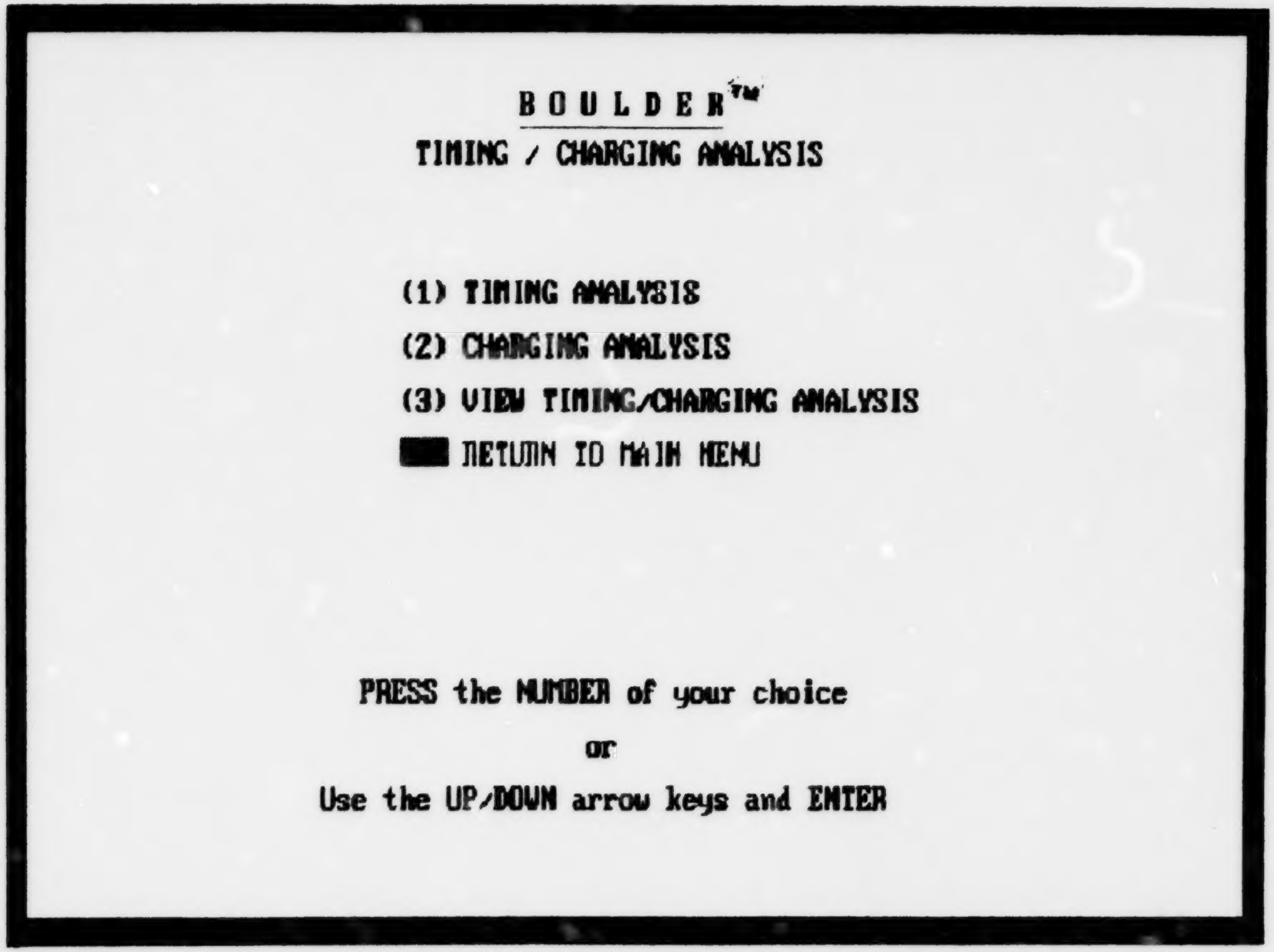


BOULDER™
DAMAGE ANALYSIS

(1) BLAST DESIGN DAMAGE
(2) EMPIRICAL VIBRATION ANALYSIS
█ RETURN TO MAIN MENU

PRESS the NUMBER of your choice
or
Use the UP/DOWN arrow keys and ENTER

Figure 11



```
                 BOULDER™
        TIMING / CHARGING ANALYSIS

   (1) TIMING ANALYSIS
   (2) CHARGING ANALYSIS
   (3) VIEW TIMING/CHARGING ANALYSIS
       RETURN TO MAIN MENU

      PRESS the NUMBER of your choice
                    or
  Use the UP/DOWN arrow keys and ENTER
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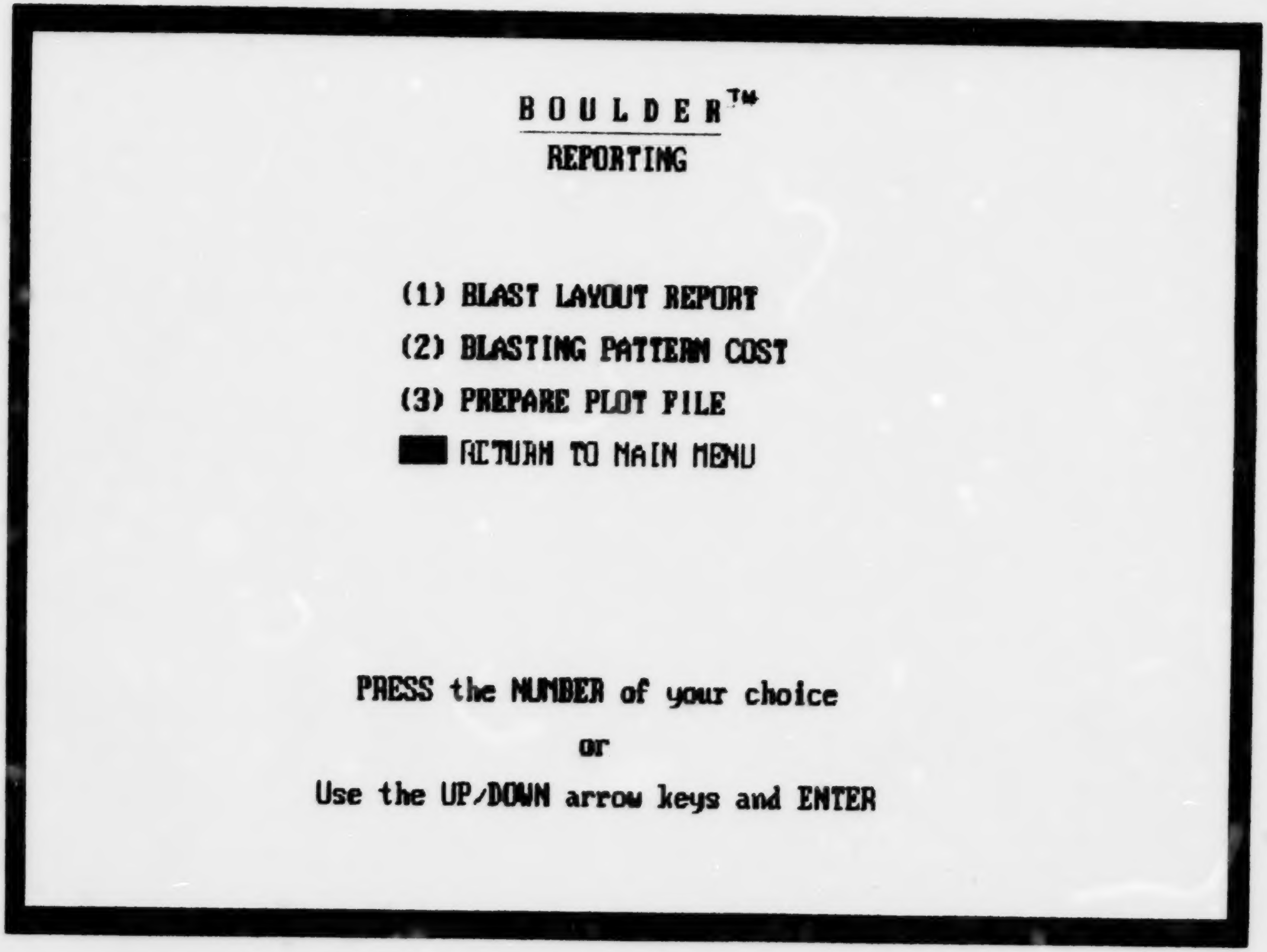
Figure 12



BOULDER™

REPORTING

(1) BLAST LAYOUT REPORT

(2) BLASTING PATTERN COST

(3) PREPARE PLOT FILE

RETURN TO MAIN MENU

PRESS the NUMBER of your choice

or

Use the UP/DOWN arrow keys and ENTER

Figure 13

SUMMARY OF INITIATION TIMES, VBM STOPE BLASTS

Delay	Nonminal Firing Time(ms)	Blast # (ms)	Blast #2 (ms)	Blast #3 (ms)	Blast #4 (ms)	Blast #5 (ms)	Blast #6 (ms)	Blast #7 (ms)	Blast #8 (ms)	Blast #9 (ms)	Blast #10 (ms)	Mean (ms)	Standard Deviation	Mean Error (%)
1	30							36		44	39	39.7	4.0	+32.3
2	50						55	59		63	54, 57	57.4	3.8	+14.8
3	75						78			80	80, 83	79.7	2.8	+6.3
4	100					113	109	102		123	113	112.0	7.8	+12.0
5	128					151	131	129		150	145, 148	142.3	9.7	+11.2
6	157					184	188	167		182		174.7	9.6	+11.3
7	190	251, 258	186	176	210	215		207	215	209	207, 210	213.0	24.0	+12.1
8	230		227		250	248	244	245	243	253	228	243.0	11.4	+5.7
9	280	291, 298	283	292	334	310	289	290	302			288.9	23.3	+3.2
10	340	360	381	374	382	380	350		378	361	341	365.2	18.4	+7.4
11	410	407	418			433	421	416	414	416	417, 449	421.2	12.5	+2.7
12	490	508, 514	517	515	503	520	522		503	516	525, 534	516.1	9.3	+5.3
13	570	602,607, 615	610	585, 610	607	633	579	570	613	616	611	604.5	17.0	+6.1
14	650	710,736	701	699	701	675	697	693	708	682		698.1	16.7	+7.4
15	725		741	778	744	734		753	753	789		752.8	18.2	+3.8
16	800		813,822	819,827	842	833			841	834		828.9	10.4	+3.6
17	875			906	918	920		852	926			904.2	30.2	+3.3
18	960			1043	1024	1027		925	981,989, 1001			988.5	38.3	+6.1
19	1025				1072	1104			1072,1078			1081.6	15.4	+5.5
20	1125				1175,1195	1231		1182	1173,1185			1186.9	24.4	+5.5
21	1225				1317	1272						1294.6	31.7	+5.7
22	1400	1462,1508			1505	1474			1338,1483			1461.7	63.1	+4.4
23	1675	1792			1794				1782			1789.3	6.4	+6.8
24	1950				1995,1999							1997.0	2.8	+2.4
25	2275	2321,2495 ,2546										2454.0	118.0	+7.9

Table 1

C.I.L. ROCK PROPERTY RESULTS

Test #	Rock Type	Density g/cc	Cp km/s	Cs km/s	PR	YM GPa
1	Hw Schist,TPSN,1400L, 3 Shear,Blk 10	2.64	5.959	3.645	0.20	84.3
2	Ore Zone Peridotite, 1400L,3 Shear,Blk 10	2.85	3.978	2.366	0.23	39.1
3	Ore,1400L,95,Blk 9	2.94	5.263	3.341	0.16	76.3
4	Schist,2800L,890,HW,Blk 5	3.04	4.077	2.671	0.12	48.7
5	Ore,2800L,890,HW,Blk 5	3.95	4.237	2.464	0.24	59.7
6	Ore,1600L,340,Blk 3	3.02	5.091	-	-	-
7	Hw Schist,2800L,30,Blk 2	2.71	4.104	2.525	0.20	41.2
8	Ore,2800L,30,Blk 2	4.01	4.305	2.701	0.18	68.8
9	Ore,1600L,340,Blk 3	2.69	5.067	3.167	0.18	63.7
10	Ore,Massive Sulphide, 1400L,3 Shear,Blk 10 Low Mineral Content	2.78	4.796	3.184	0.11	62.3
11	Ore,Massive Sulphide, 1400L,3 Shear,Blk 10 High Mineral Content	4.61	2.861	1.609	0.27	30.3

Table 2

ROCK MASS INPUT

Rock Type Name

Rock Density

Uniaxial Compressive Strength

Rock Quality Designation 'RQD'

Rock Quality 'Q'

Rock Mass Rating 'RMR'

Kuz-Ram Rock Factor

Discretionary Vibration Constant

Mean Spacing Joint Set #1

Mean Spacing Joint Set #2

Mean Spacing Joint Set #3

Mean Joint Length Set #1

Mean Joint Length Set #2

Mean Joint Length Set #3

Dip Joint Set #1

Dip Direction Set #1

Dip Joint Set #2

Dip Direction Set #2

Dip Joint Set #3

Dip Direction Set #3

Table 3

EXPLOSIVE TYPE INPUT

Explosive Name

Cartridge or Powder

Diameter of Cartridge

Density of Explosive

Relative Weight Strength

Borehole Pressure

Velocity of Detonation

Cost (Cartridge or Powder)

Cost of Detonators

Cost of Detonation Cord

Cost of Primers

Table 4

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ANNUAL PROGRESS REPORT 87/88 B1 TO B32

APPENDIX A

LITERATURE REVIEW/FINAL REPORT

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RESEARCH & DEVELOPMENT IN MANTIOBA MINES

TASK 5 - OPTIMIZATION OF BLAST DESIGN FOR BLASTHOLE MINING IN MANITOBA MINES

LITERATURE SURVEY/FINAL REPORT

Prepared For:

CANADA CENTRE FOR MINERALS AND ENERGY TECHNOLOGY
(CANMET)

By

INCO LTD., MANITOBA DIVISION, THOMPSON

Work on the project was conducted under the auspices of the Canada Centre for Mineral and Energy Technology, Energy, Mines and Resources Canada under the Canada/Manitoba Mineral Development Agreement.

DSS File No. 14SQ.23440-5-9251
DSS Contract Serial No.: 23440-5-9251/01-SQ
CANMET Project No.: 5-9251
CANMET Scientific Authority: J. Pathak

1 September 1986 - 31 March 1987

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1.0 INTRODUCTION

The primary objective of the CANMET task 5 is to develop blast design software which meets the following criteria:

- easy to use by the engineers and first line supervisor
- makes a good first estimate of a suitable blast design
- capable of meeting day-to-day production requirements:
  - checking actual blast design
  - checking timing and charge weights
  - produces drilling, charging and timing reports in a form suitable for the operating personnel
- recording and filing of blasting records and layouts

In essence the software should meet daily design requirements and be a'tool' to improve design and operating procedures. It should be updated with experience on a regular basis. The strong empirical base should overcome the necessity for sophisticated modelling methods which accurately reflect the blasting process. An accurate reflection may, in any event, be illusory at this stage due to the inhomogeneity inherent in any rock mass. It is interesting to note that all the most recent proprietary software from explosive suppliers relies on calibration blasting. In addition none of the available software would meet the day-to-day production requirements.

The objective of this literature survey is to present a brief review of blasting especially those models which are used to describe breakage and which form the basis of the available blast design software. The conclusions from the literature survey will be used to formulate a design flow path for this project and to constructively criticize commercial software.

The literature review should be the foundation on which the project can grow. It is not exhaustive but does present a fairly complete picture of blasting research.

The review is set out under three headings:

- breakage
- fragmentation
- damage

"Breakage", "fragmentation" and "damage" are all interconnected. Breakage is effective when the in situ material is displaced into a muckpile with a certain fragmentation. Damage is the permanent disturbance of the material that remains - the material not sufficiently fragmented to be moved into the muckpile.

The blasting process is complex. The violence of the blast is so catastrophic that monitoring or observation of the immediate activity around the borehole is impossible. The result is that theories are used to describe the breakage rather than actual results and it is difficult to substantiate these theories. For this reason it has proved more convenient, although basically incorrect, to divide the breakage process into the three separate relationships.

2.0 BREAKAGE

A number of processes or models have been used to describe and explain breakage by explosives:

- Cratering;

originating with the Livingston theory (1) and extended into a viable mining method as Vertical Crater Retreat (2). The Livingston theory provides ..." a means of correlating impact and explosion so as to identify ranges of similar mechanical behaviour of materials and of relating the sequence in which they develop to the non-separable and interdependent fundamental quantities energy, mass and time ....."

In essence crater theory is used to extrapolate the breakage from a single test with a relatively small hole to a pattern of large holes. It has also been used to describe the breakage at the collars of holes and also at the toe for surface blasting.

ₒ Strain;

when the explosive detonates the very high pressures which impact the borehole wall create high stresses and a resultant radial strain. The strain causes cracks to form through tensile failure. The radial strain attenuates very quickly around the borehole (4) and the frequency of cracking therefore reduces. At the same time the gas pressure reduces and reaches a level where it cannot move the cracked material into a muckpile. There will be a certain critical burden at which the numbers of cracks and the gas pressure are insufficient to result in a reasonable muckpile. The "strain" model directly relates the "energy" of the explosive to certain rock properties to produce a rupture envelope around a particular hole. This "rupture" envelope can be used to optimize the drilling pattern.

ₒ Empirical;

a relationship which includes the factors which influence the breakage process:

> breakage = f(rock. explosive energy. burden/spacing. degree of breaking difficulty. etc.)

which is calibrated against many full scale blasting experiments to define a set of constants which provides a "best fit" of the data and constitutes a logical experience model. This method can be expanded to include a number of rock properties which best describe the resistance to rupture of the rock mass and this resistance can be expressed as a powder factor, or specific charge. It remains basically an empirical method which assumes fairly average conditions for charge calculation and then distributes the specific charge through the rock mass dependent on geometry.

The above three broad descriptions are adequate to cover most of the relevant blast design literature. Before detailed discussion of these "breakage" models it is of benefit to describe the detonation of an explosive within a rock mass. This will put the model assumptions into perspective, or reality, and enable constructive criticism.

The detonation of an explosive is extremely fast and very high gas pressures are reached. These gas pressures are sufficient to cause crushing in the vicinity of the borehole as they overcome the intact strength of the rock. Figure 1. The stress wave radiates out from the blasthole and produces radial cracks due to the "hoop" stress component. The intensity of the stress wave falls rapidly further from the hole. It is suggested that the radial cracking will continue as long as the tangential tensile stress at the crack tip exceeds the tensile strength of the rock (5). The frequency of cracks decreases with distance from the borehole.

The gas pressure generates the dynamic stresses. It also promotes the progression of the cracks by streaming into the cracks and extending them by acting on the sides of the cracks and forcing them open.

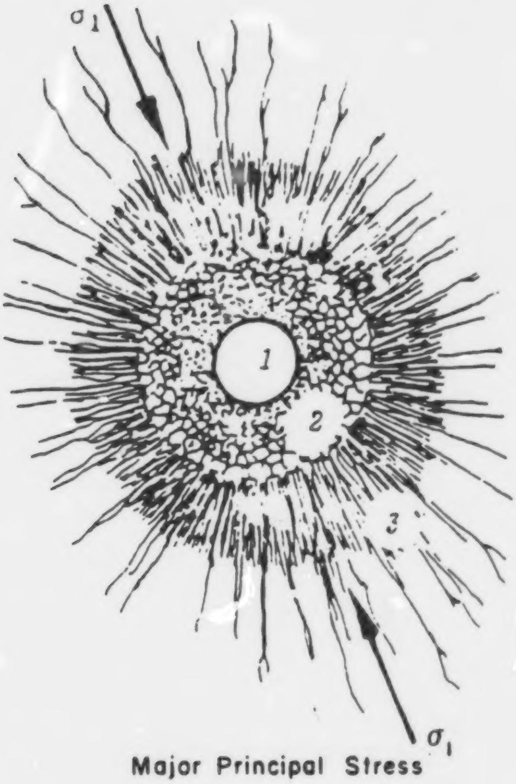

σ_1
1
2
3
σ_1
Major Principal Stress

FIGURE 1. Idealized picture of fracturing induced by detonation of an explosive in a borehole.

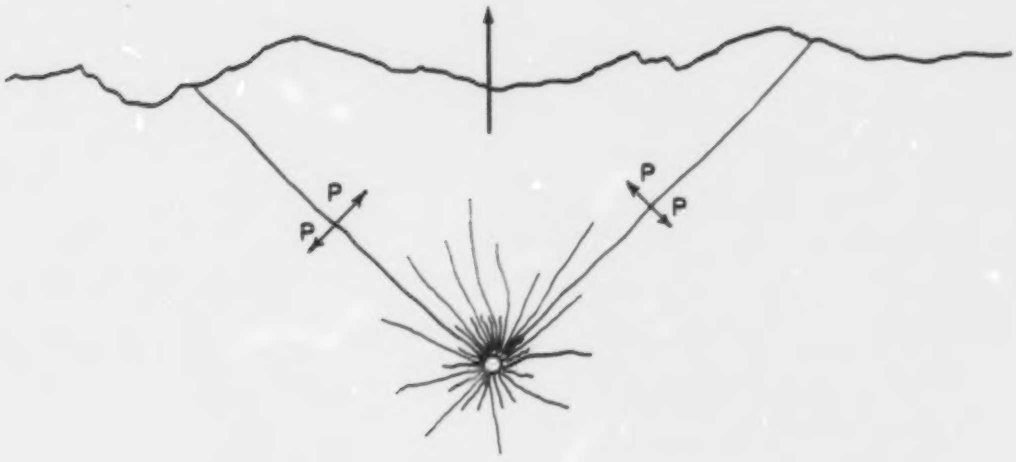
1. Borehole
2. Pulverized zone
3. Radial cracks with preferential growth parallel to σ_1

Fragments are detached from the body of the mass if the cracks extend to a free face and there is still sufficient gas pressure to move the fragments, figure 2.

The cracking process is complicated by the reflection of the radial stresses from free surfaces but the action of these reflected waves does not appear to be significant.

The above explanation remains the standard description of the process of rock breakage. There is considerable argument regarding the relative contribution of the shock, or stress wave, and the gas pressure to the breakage process. But the overall model remains the same and the theoretical considerations are confined to cracks in intact material.

The rock mass is far from intact. It is traversed by a pattern of discontinuities such as joints, bedding planes and foliation surfaces. In fact the rock is already broken to varying degrees depending on the competence of the rock mass. It can be argued that the cracks in intact material originating from the borehole tend to terminate at prominent existing cracks, as illustrated in figure 3. The breakage process due to "strain", therefore, might be very limited depending on the hole spacing and the joint frequency.



PLAN VIEW

P = Gas Pressure applied to
Crack Surface

FIGURE 2. MECHANISM OF "HEAVE" WHEN CRACKS HAVE REACHED THE LIMITS OF THE BURDEN.

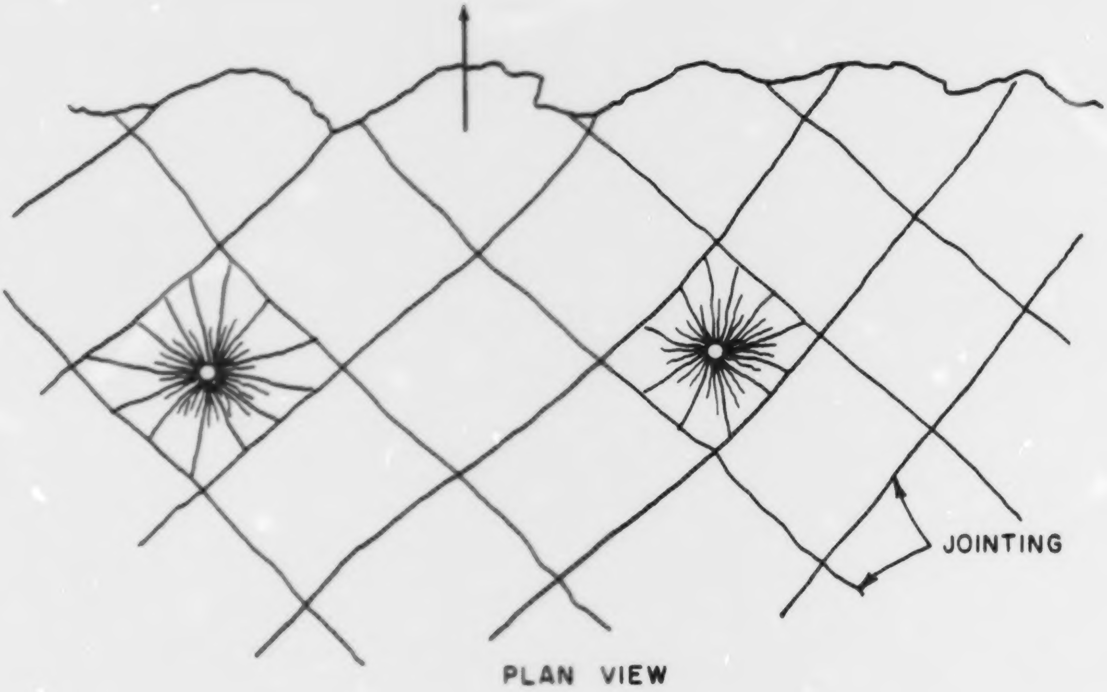

JOINTING
PLAN VIEW

FIGURE 3. FRESH CRACKING CONFINED TO BLOCKS OF ROCK INTERSECTED BY THE CHARGED HOLE.

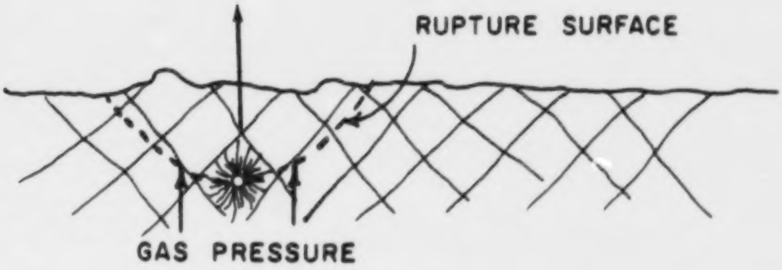
There are other processes of breakage which are valid but which have not been modelled and there is virtually no description in the literature:

- Tearing and rupture of the material as the burden is displaced by the gases acting on the new and existing crack surfaces and forcing the material out figure 4(a). The amount of "rupture" will depend on the jointing frequency and how tightly the blocks are held in the rock mass.

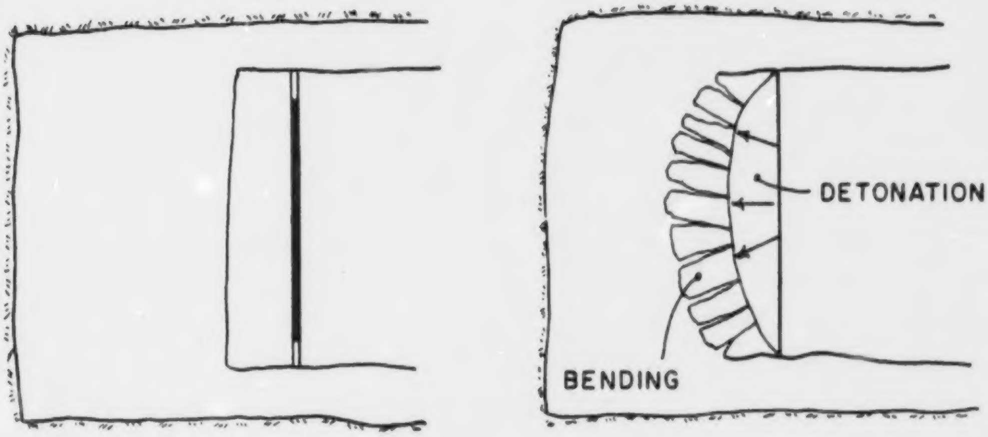
- Bending of the burden in front of a line of drill holes, figure 4(b). The "tearing" and "rupture" will include a bending element, again due to the gas pressure. This is well documented from high speed photographs. The amount of bending will depend on the ratio of burden to charge length and will probably be at a minimum for spherical charges.

- Impact of blocks against each other and with the walls of the stope, figure 4(c). The velocity of the blocks depends on the gas pressure still remaining once the burden has loosened. The velocity is fairly low and the amount of "intact" failure in hard rock is unlikely to be significant but blocks which contain tight discontinuities may be further fragmented.

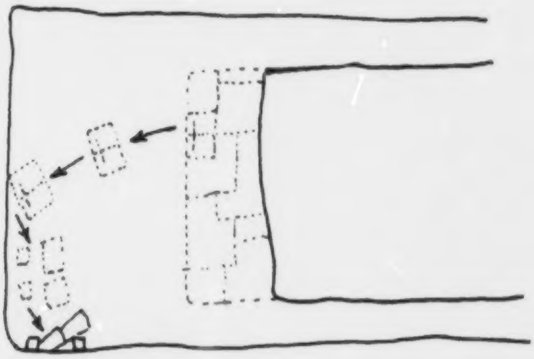
Both gas pressure and shock play a part in the breakage process but it is difficult, if not impossible, to apportion their individual contributions, although it is generally accepted that in a "rock mass" gas, or heave, is the more important factor.


RUPTURE SURFACE
GAS PRESSURE

(A) GAS PRESSURE WITHIN FRESH CRACKS AND WITHIN EXISTING CRACKS TEARS OUT THE BURDEN.


DETONATION
BENDING

(B) BENDING OF THE BURDEN IN FRONT OF A DRILL HOLE.



(C) IMPACT

FIGURE 4. BREAKAGE PROCESSES OTHER THAN STRAIN.

The above discussion sets the scene for an appreciation of the different breakage models.

2.1 Crater Theory

The definition of crater blasting is best left to its main proponent (3) and the following is quoted.

"....A crater blast is where a spherical charge is detonated beneath a surface that extends laterally in all directions beyond the point where the surrounding material would be affected by the blast.

In analyzing crater blasts, it has been found that there is a definite relation between the energy of the explosive and the volume of material that is affected by the blast, and this relationship is significantly affected by the placement of the charge.

Livingston determined that a strain-energy relation exists, expressed by an empirical equation:

$$N = EW^{1/3}$$

where N is the critical distance at which breakage of the surface above the spherical charge does not exceed a specified limit, E is the strain energy factor, a constant for one given explosive-rock combination, and W is the weight of the explosive charge.

The same equation may be written in the form of:

$$d_b = \Delta E W^{1/3}$$

where d_b is the distance from the surface to the center of gravity of the charge, i.e., depth of burial, and Δ equals d_b/N which is a dimensionless number expressing the ratio of any depth of burial compared to the critical distance. When d_b is such that the maximum volume of rock is broken to an excellent fragment size, this burial is called optimum distance (d_o).

An actual cratering experiment consists of a number of cratering-type shots in which the type of explosive, its charge weight, and the rock are constant and only the depth of burial varies.

Critical distance is obtained by observation. All craters are excavated and their volume determined. Fig. 5 shows the plot of data obtained from a cratering test in which the depth ratios ($\Delta = d_b/N$) are plotted against energy levels (volume/charge weight: V/W). This idealized curve of Livingston also shows the important transition limits from the isolation range to the other extreme which is the air-blast range. Craters may vary considerably, Figure 6.

Once having this curve established, by taking the appropriate value from the corresponding part of the curve and applying the generally accepted system of cube root scaling, production-scale blasts can be designed for spherical charges to satisfy any demand from simply

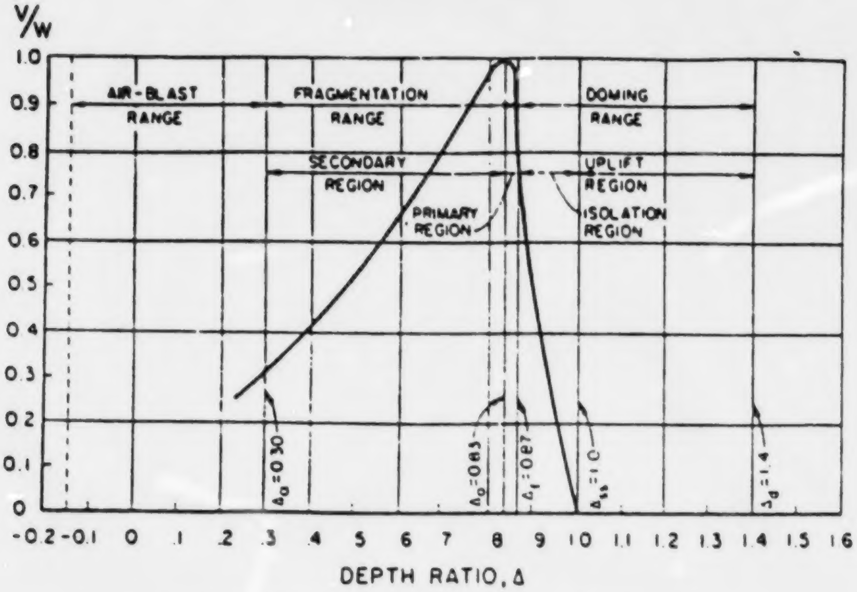


FIGURE 5. CRATERING TEST IN WHICH DEPTH RATIOS ARE PLOTTED AGAINST ENERGY LEVELS. THIS IDEALIZED CURVE OF LIVINGSTON ALSO SHOWS THE IMPORTANT TRANSITION LIMITS FROM THE ISOLATION RANGE TO THE OTHER EXTREME WHICH IS THE AIR-BLAST RANGE.

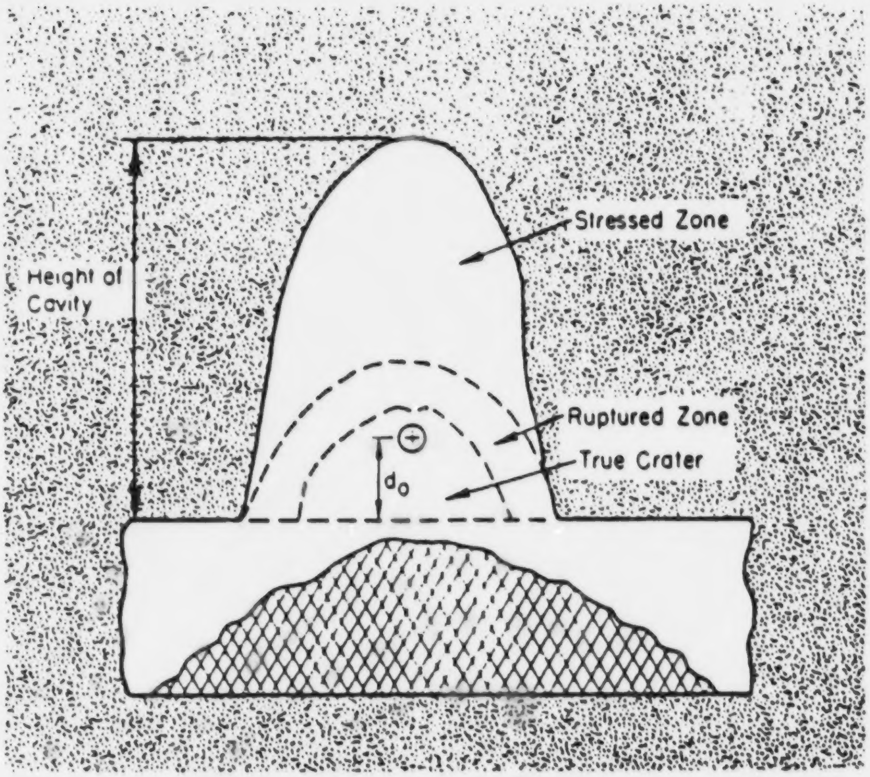

Height of Cavity
Stressed Zone
Ruptured Zone
True Crater
d_0

FIGURE 6. DEPENDING ON ROCK PROPERTIES AND STRUCTURAL GEOLOGY THE TOTAL HEIGHT OF THE CAVITY MAY EXCEED THE OPTIMUM DISTANCE OF THE SPHERICAL CHARGE FROM THE BACK MANY TIMES.

isolating the rock from its surroundings to the effective breaking of the rock, or even for casting with explosives......"

The strain energy relationship is a law of conformity. It makes no assumption regarding rock strength, structure or explosive energy other than to assume that the interaction of particular explosive in a particular rock can be described by a single constant. The relationship is therefore a vehicle for transferring experience but only in the same rock. There are no functions which take into account the rock strength, the influence of structure or the variation in explosive strength.

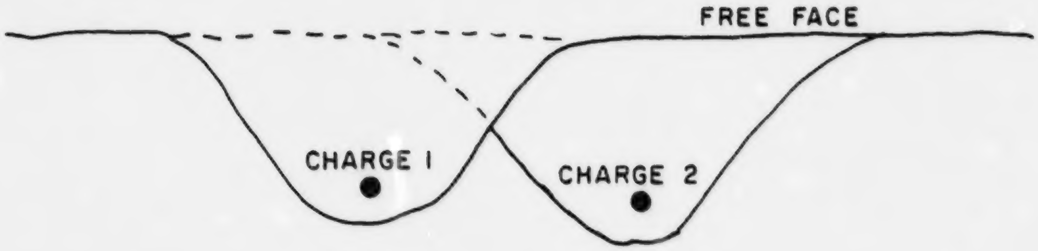
It is suggested by Livingston (6) that these crater tests should be as close as possible to the charge weights and bench geometry used in practice. This suggestion recognizes the limitations of the relationship. It can only be used if cratering design data is available. The cratering tests are assumed to be unique for an explosive rock mass combination. It must also be unique to the interaction of geological structure. The smaller the charge used in an experiment the less the structure will influence the results and the more the results will be dependent on the intact properties.

A further fundamental flaw in the procedure is that no interaction effect between adjacent craters is recognized. Only the first crater blast is representative of the crater experiments and subsequent charges do not break to a single free face but to a combination of free faces, figure 7. The "stressed" zone in Figure 6 will change fundamentally dependent in whether there is a single crater or a combination of craters.

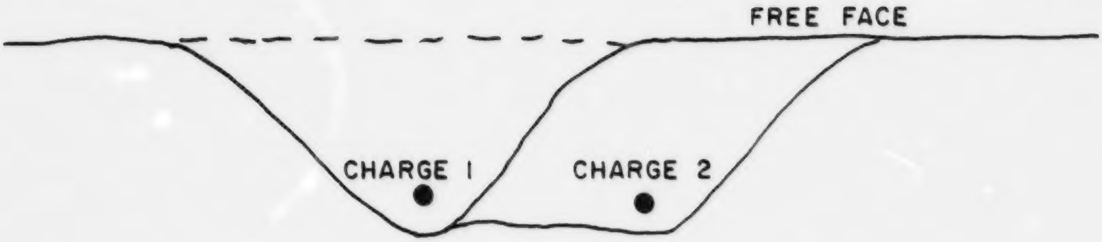
The interaction of craters has a further important effect. A single hole cannot be advanced by a series of crater blasts unless the back of the crater breaks to a size and shape which would accommodate the next crater. If not, it will very quickly choke. This argument can be extended. The wider the orebody the more likely it is that a combination of craters will break back to a flat surface providing the next level of charges with a reasonably flat free face.

The crater design relationship describes a unique relationship of rock and explosive interaction. If the above discussion is a reasonable assessment of the factors which affect the breakage then even if the rock mass, charge weight and explosive type remain the same there will be a very large degree of uncertainty regarding the actual breakage. This uncertainty can be illustrated from actual blast profiles in different stope geometries using, essentially, the same blast parameters of hole size and charge weight, Figure 8.

Edwards (7) goes further".....Site specific geology plays a major role in the final shape of a crater. Sites with a minimum of flaws and fractures (pre-shot) are symmetrical and reproducible from experiment to experiment. Sites with complex pre-shot flaw distributions tend to be asymmetric and the character of the


FREE FACE
CHARGE 1
CHARGE 2

GENERALLY ACCEPTED


FREE FACE
CHARGE 1
CHARGE 2

ACTUAL CASE

FIGURE 7. COMPARISON BETWEEN THE GENERALLY ACCEPTED DESIGN PRACTICE AND AN ACTUAL CASE. SHAPES WILL VARY DEPENDENT ON THE ROCK MASS STRUCTURE. (Mining Resource Engineering Ltd. Report, June 1986 CANMET Project No. 5-9086).

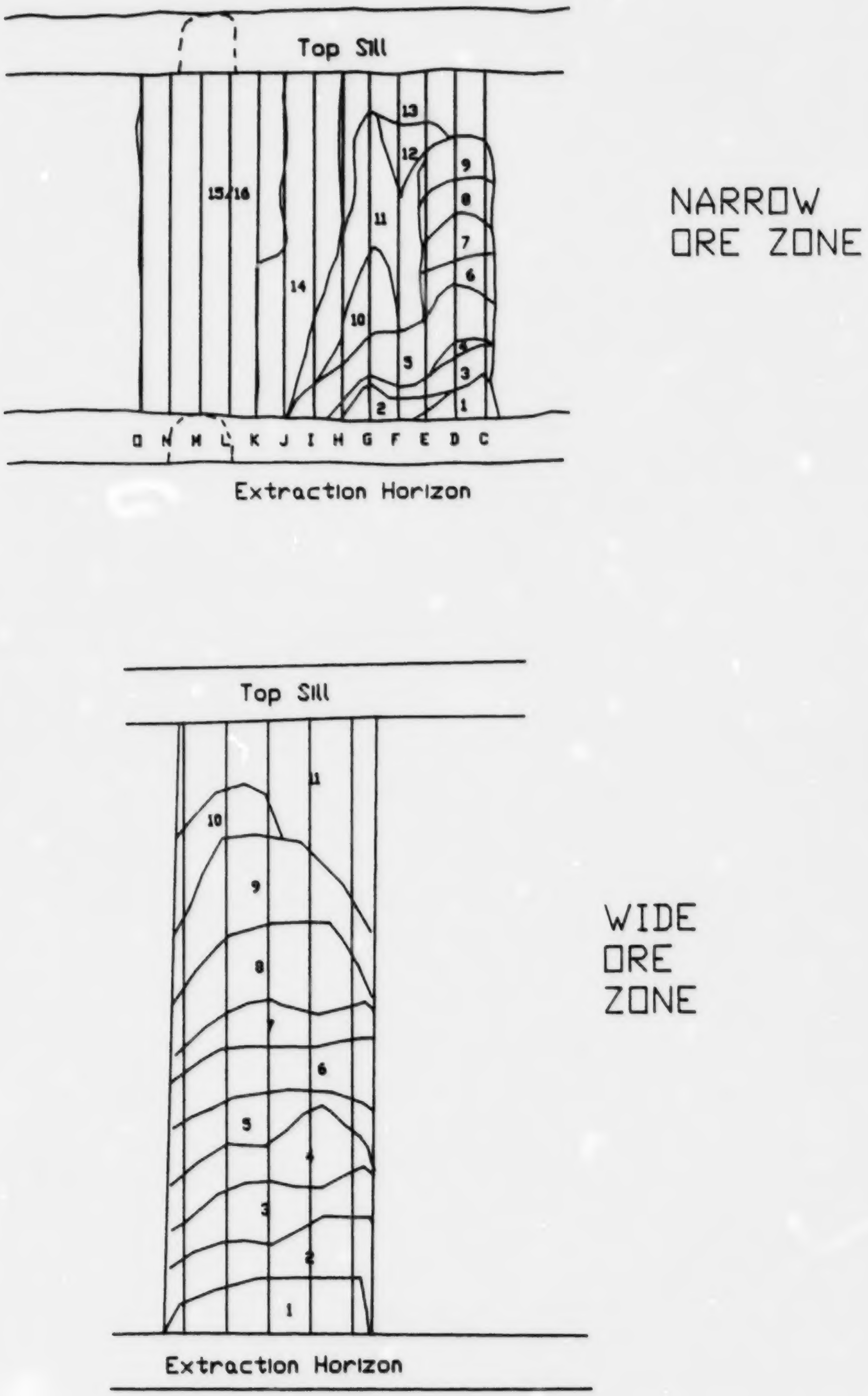

Top Sill
13
12
9
8
15/16
11
7
6
14
10
5
4
3
2
1
O N M L K J I H G F E D C
Extraction Horizon
NARROW ORE ZONE
Top Sill
11
10
9
8
7
6
5
4
3
2
1
Extraction Horizon
WIDE ORE ZONE

FIGURE 8. Variation in Practice.

craters varies substantially from experiment to experiment...".

The cratering relationships are relatively simple but their implementation is cumbersome and the results are extremely variable.

2.2 Strain

The strain model can be better defined as a "rupture" envelope around the detonating borehole, Figure 9. The "rupture" has to be sufficiently intense and the remaining gas pressure sufficiently high for the material to be displaced. This type of model appears to be the basis for some of the proprietary software packages and as such the literature is fairly limited. A representative description of a strain model is given by Harries (8).

"...The strains produced in rock by blasting can be calculated by assuming that the blasthole is a thick-walled cylinder of rock subjected suddenly to an internal pressure equal to the pressure the explosive would have if it reacted in its own volume. In order to calculate the strain, Young's modulus and Poisson's ratio of the rock have to be known.

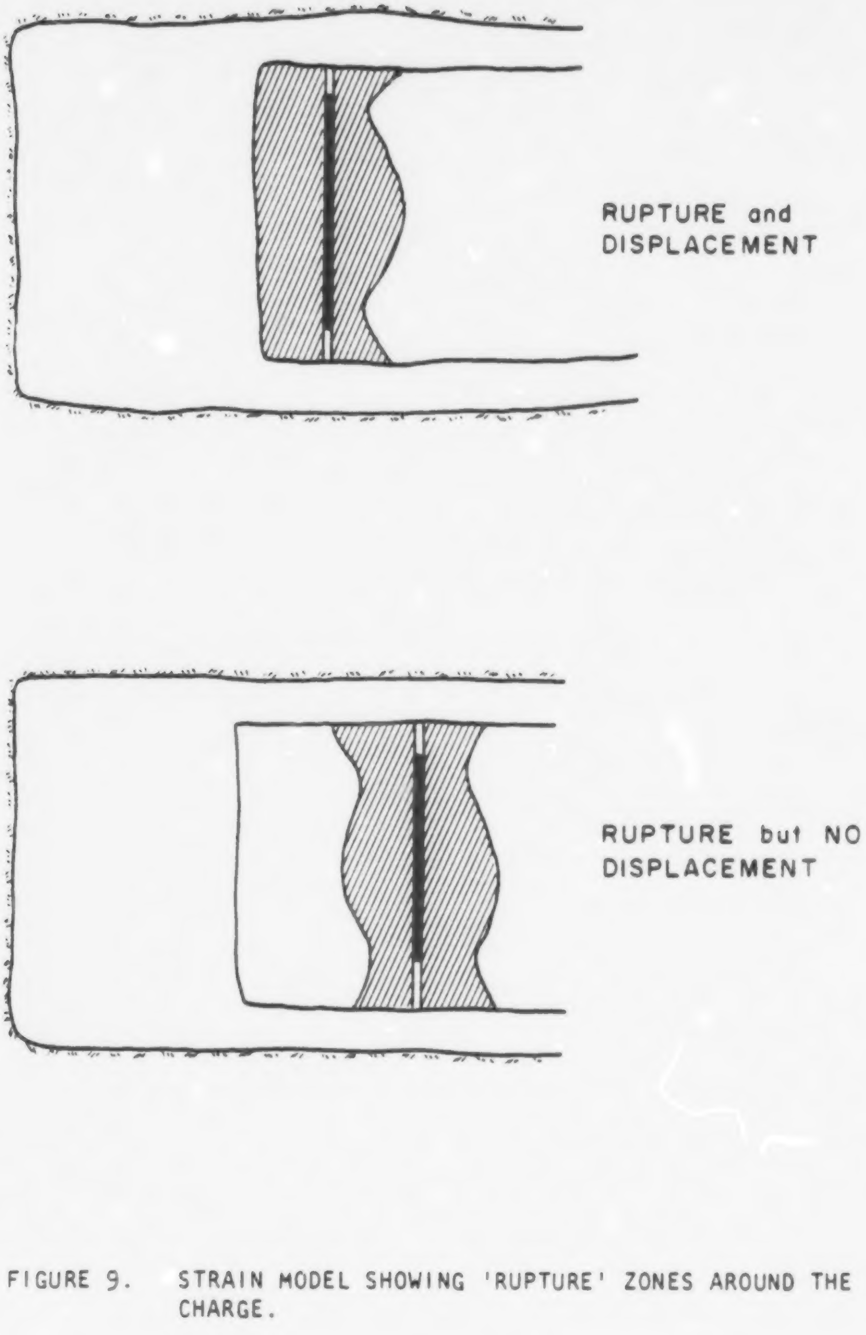

RUPTURE and DISPLACEMENT
RUPTURE but NO DISPLACEMENT

FIGURE 9. STRAIN MODEL SHOWING 'RUPTURE' ZONES AROUND THE CHARGE.

The tangential strain ε generated by an explosive charge around a blasthole radius b at a distance R from the blasthole has been shown to be

$$\varepsilon = (Kb/R \exp(-\alpha R/b)$$

where K is the strain at the blasthole wall and α is the strain absorption co-efficient.

If the rock has a tensile breaking strain T the number of radial cracks N which will be found on the circumference of a circle radius R whose centre is the blasthole is

$$N = (N_0 b/R) \exp(-\alpha R/b)$$

where No = K/T = number of cracks at the blasthole wall.

The strain at the blasthole wall K can be found from the elastic rock properties and the explosive properties. The strain absorption co-efficient has been found experimentally to vary from 0.002 to 0.008. A typical graph of crack length versus number of cracks of that length is shown in Figure 10........"

Input data can vary from static intact rock properties such as Young's modulus, poisson's ratio and compressive strength to dynamic values of compressive and tensile strength and values of "p" and "s" wave velocities as seismic values of the rock mass.

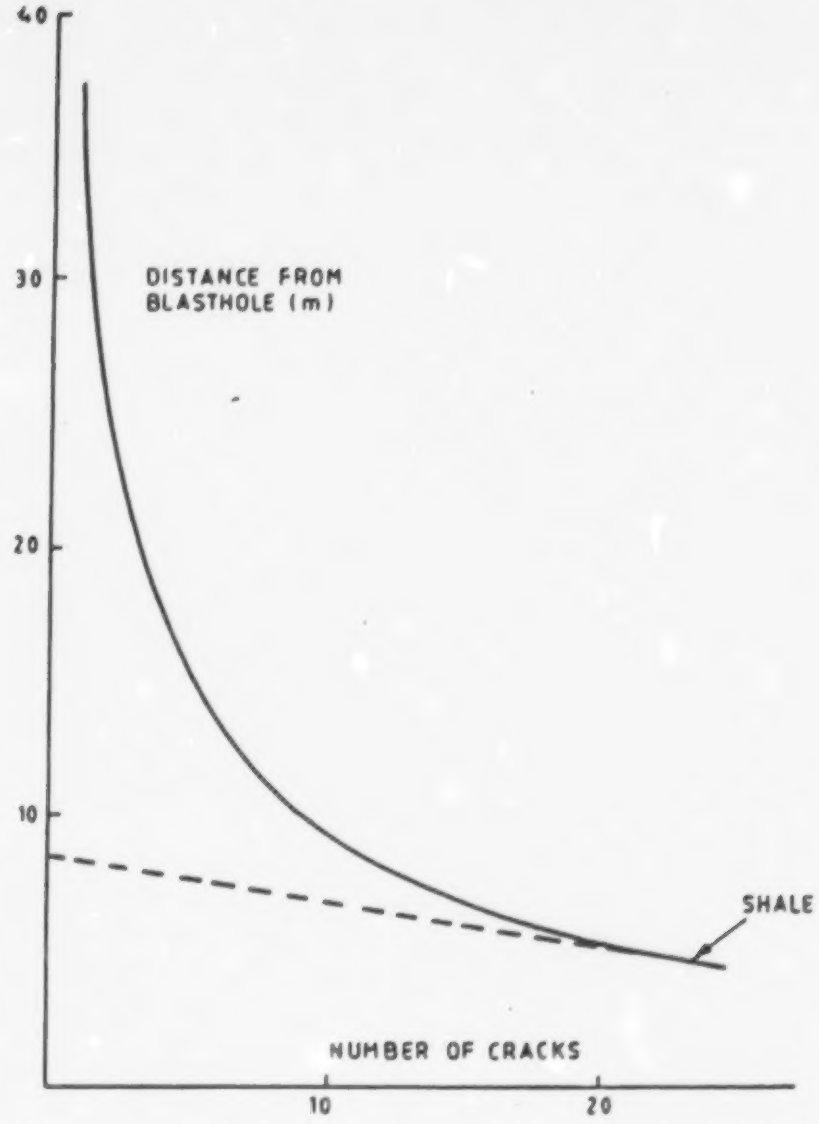


FIGURE 10. CRACK INTENSITY VERSUS CRACK LENGTH

The use of the strain model in practice relies on calibration in the recognition that the input data is insufficient to describe the inherent variability of the material being blasted. The model goes much further than simply being a vehicle for the transfer of experience. It recognizes and simulates a fracture process and includes an estimate of available energy dependent on the explosive used and then develops the rupture envelope according to rock strength.

What it fails to recognize at present, is that a rock mass is already "ruptured" therefore the contribution of the fresh cracking will vary considerably dependent on the rock mass competency. The only rock mass properties usually included in an estimate of breakage are the seismic velocities and these are extremely poor estimates of rock mass competency.

In essence, the relationship estimates the energy source and the reduction in energy with distance from the borehole. This reduction in energy coupled with an estimate of the rock resistance to failure, as disturbance, defines the limiting burden beyond which breakage may occur but with little or no displacement of the broken rock.

The criticism is that the intact rock properties and seismic velocities are not necessarily the most appropriate measures of the rock mass resistance to displacement.

2.3 Swedish Blasting Method

This is essentially an empirical method but is separately described as it is the most successful and enduring of all the blasting models which have been developed. The method is based on the assumption that there is a minimum charge required to loosen the rock (9). It then considers the influence of faulty drilling, throw and swell, fragmentation and the effect on the surroundings. The method was devised from systematic blasting experiments carried out according to theoretical assumptions (9), the starting point being to group the relevant functions:

$$Q = f_1(V,K,E,h,d,s,p,u.c_i,)$$

Where Q = amount of charge required for tearing the burden

V = maximum burden

K = bench height

E = spacing between holes

h = charge height

d = hole diameter

s = explosive weight strength

p = explosive density

u = propagation velocity

c_i = factors depending on the rock strength and including fixation (breaking difficulty) and swell.

Experiments to determine f_1 were carried out with:

$$Q = f_1(V,K,h,d) \quad (s\ p\ u\ c_i \text{ held constant})$$

The relationship can further be reduced until V is the only variable giving the following relationship for ordinary Swedish bedrock:

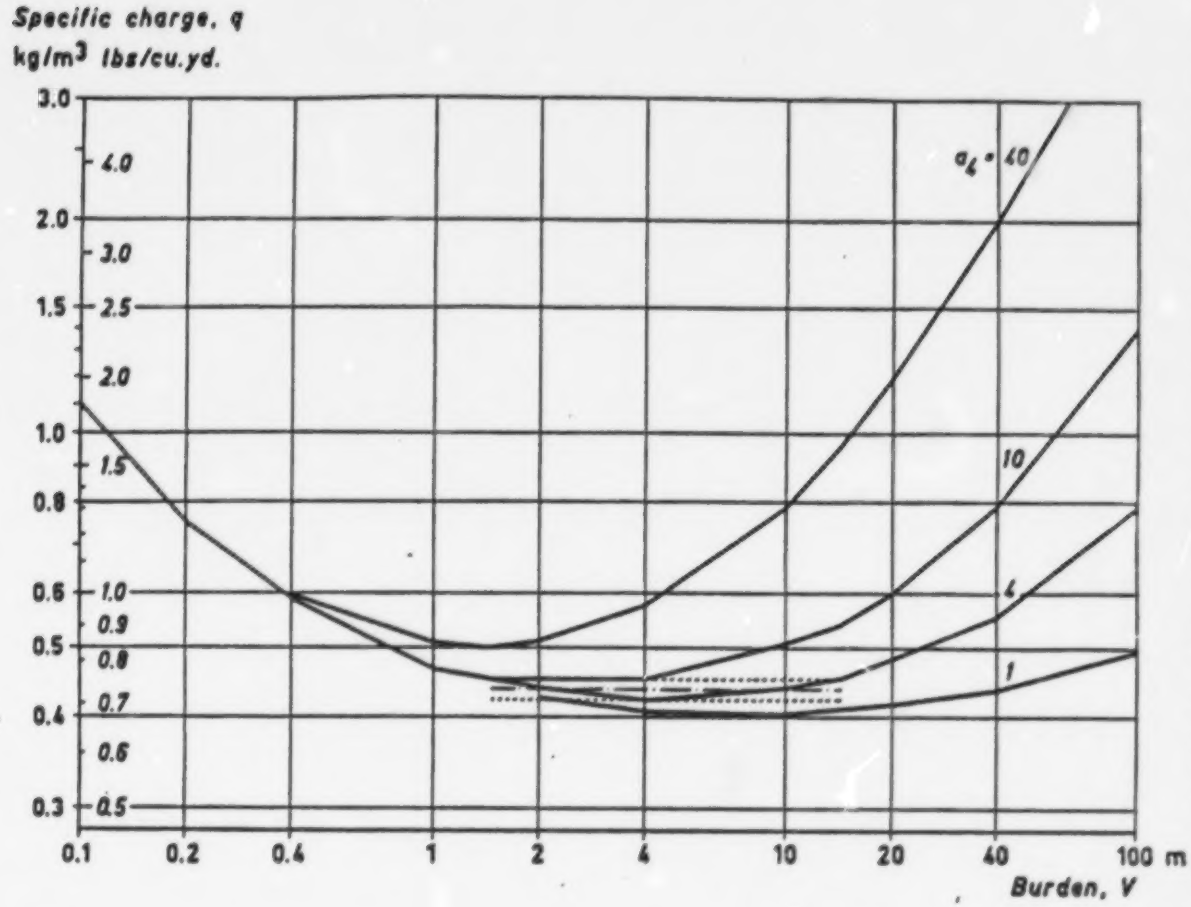
$$Q = 0.01V^2 + 0.40V^3 + 0.004V^4$$

The relationship is graphically displayed in Figure 11. This is fundamental to the Swedish system since it suggests that a specific charge, $V = 0.4 - 0.5$ kg/m^3, can be used to describe most blasting situations. The implication being that rock strength and quality have little influence on the maximum burden which can be broken by a certain charge.

Blasting relationships are then developed on the following basis:

- toe charge: controls the maximum burden that can be broken. The charge must have a certain total mass and mass per meter. The toe charge is very confined and has the maximum breaking difficulty of the whole charge column and therefore has the maximum specific charge.

- column charge: the rest of the charge, defined as the "column", is factored down from the toe charge to account for the less confined breaking geometry.



Charge in kg/m^3 (lb./cu.yd .) in conformal scaling with a swell component of a_4 = 40, 10, 4 and 1.

FIGURE 11. BASIS FOR STANDARD SPECIFIC CHARGE IN SWEDISH BLASTING METHOD.

The modelling takes into account the following influences on the maximum burden:

- explosive strength: compares explosives by the "relative weight strength".
- fixation: assesses the influence of geometry on breaking by taking into account the angle of the holes, number of holes in a row and whether the toe is breaking to a free bottom.
- rock constant: a single "constant" is used to characterize the rock which is in the form of a specific charge c = 0.28 - 0.35 kg/m^3 for most rocks in their experience.
- degree of packing: takes into account the "charged" density.
- swell: where the burden is loaded with previously broken rock extra charge is required to create swell space.
- drill hole deviation: correction to the maximum burden made for drill deviation.

The constants used in the various relationships have been derived empirically but the relationships have been based, to a certain degree, on theory.

The basic relationship which is developed is as follows:

$$V = \frac{d_b}{33}\sqrt{\frac{P.s}{\bar{c}f(E/V)}}$$

where V = burden, m

db = diameter of the drill bit at the bottom of the hole, mm

P = degree of packing, kg/dm^3

S = weight strength of the explosive (1.3 for blasting gelatin)

$\bar{c}$ = c + 0.05 for V = 1.4 - 15m

$\bar{c}$ = c+ 0.07/V for V <1.4m
and c = rock constant usually 0.4, kg/m^2

f = fixation factor

E/V = spacing to burden ratio

There are a number of other relationships in the Swedish Method which are used to modify V and to estimate charge weights and lengths. The above relationship is included to illustrate the simplicity of the modelling.

The "Swedish Blasting Method" has been in operation for over 25 years and has proved remarkably successful. It recognizes the complexity of the breaking process by assigning a "powder factor" as a description of the resistance to breakage, but it gives no method for assessing that "powder factor". Although simplistic it has shown itself to be as least as effective as far more complicated relationships. It can be argued that the "weight strength" method of comparing explosive strengths is not sufficiently rigorous as different energy attributes have variable effects depending on the rock mass characteristics. In addition non-ideal detonation is ignored.

2.4. Empirical

The "empirical" method as defined previously is not very different from the Swedish Method. The empiricism can be extended to cover more blasting situations such as a pure crater blasting, very small breaking faces and decked charges but the basic concept would remain. The improvement, or extension, would be in better estimating the rock constant.

The major criticism of the previous models, crater and strain, is that only intact constants are used to describe the resistance of the rock to breakage. The Swedish method goes one step further by using a single "resistance to breakage" constant which is a specific charge. It is suggested that this single constant can be estimated in the same manner that excavation stability is estimated at present, by a rock mass classification technique. At present the design powder factor concept is used by many blast design engineers. The experienced engineers will choose a design powder factor based on his own experience of the rock mass, the hole size and the geometry of the blast, the size of the free face, the burden to stemming ratio and other factors. But he may have no procedure for quantifying the influences.

It has been argued, Figure 2, that the burden can be displaced once there are cracks through to the free face in a position that frees the rock. The ease with which this will take place depends on how many cracks already exist and the condition of these cracks, the strength of the rock and resistance to new cracks, the geometry of the hole with respect to the burden and finally, the need for swell.

A "rock" powder factor will be a function of the following parameters:

- intact properties; resistance of the intact rock to creation of fresh tensile cracks and to shear.
- joint shear strength; degree of difficulty in loosening the individual blocks of rocks, or, the degree of interlocking.
- jointing frequency; a rock mass is already flawed, already has an "inherent" fragmentation, the frequency of the joints and the number of sets will also influence the resistance to loosening.
- rock density; the explosive not only loosens the rock mass but also displaces it into a muckpile. In open stoping gravity usually assists in this process.

The above parameters have been used to develop a function (10) based on previous work by Ashby (11):

$$\text{rock powder factor} = 95\ \mathrm{Ln}\left(\frac{\text{density}.\tan(\phi+i).\sqrt[3]{\text{UCS}}\ .\sqrt{\text{hole size}/100}}{(115-\text{RQD})\ /\ 3.3}\right) + 540$$

where - the rock powder factor is in g/m^3

and, density = g/m^3

ϕ and i = the base friction and roughness angles for joint surfaces

UCS = unconfined compressive strength in MPa (Tensile strength is more appropriate but is difficult to measure)

Hole size = diameter in mm

(115 - RQD)/3.3 = a measure of the joints per cubic metre of rock.

This relationship includes the hole size as an estimate of the effect of increasing the pattern with the drill holes intersecting fewer intact blocks. The above relationship is shown graphically in Figure 12.

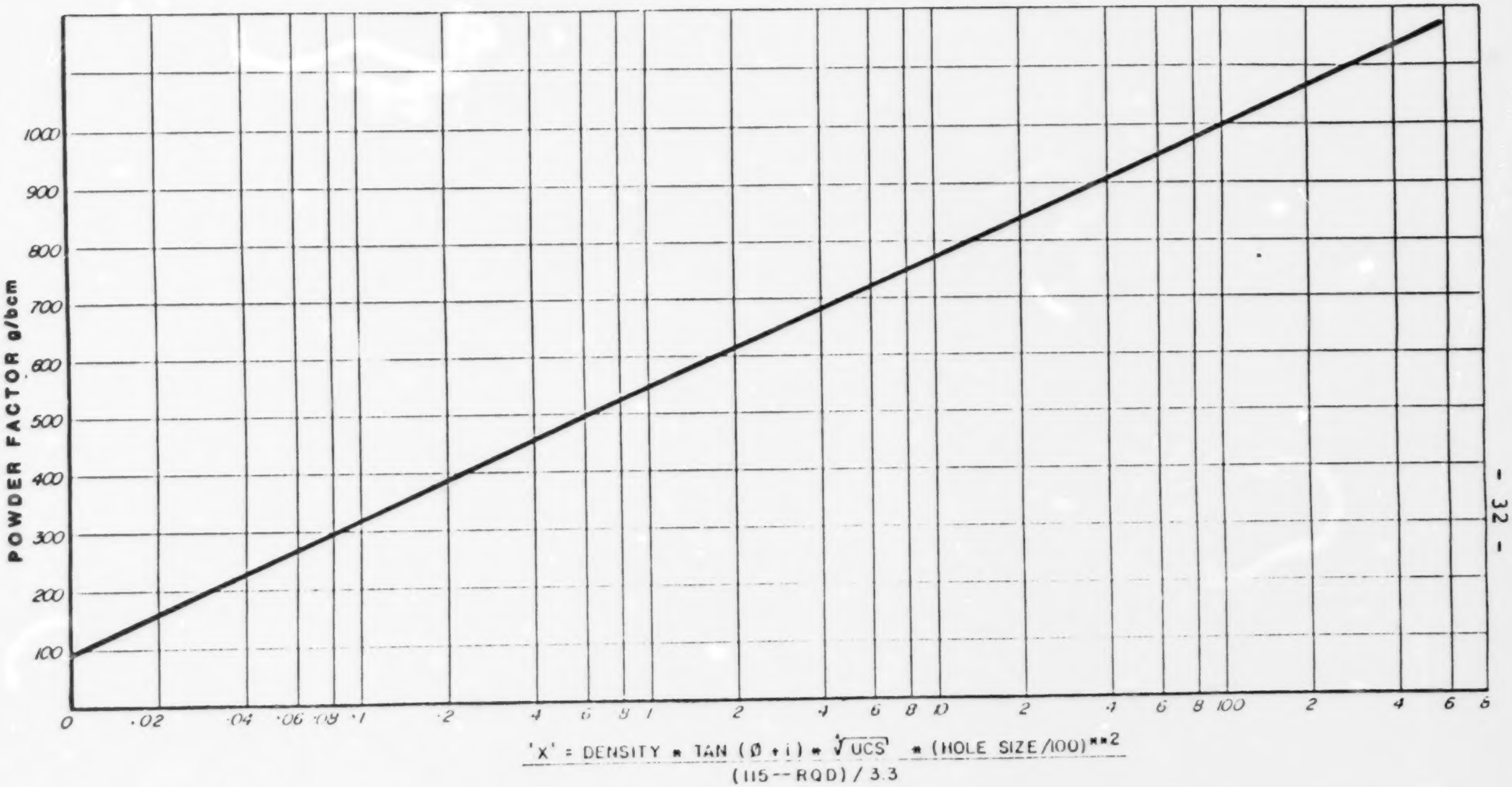


FIGURE 12. EMPIRICAL METHOD FOR ESTIMATING A 'ROCK' POWDER FACTOR.

The parameters are also included in current rock mass classification techniques and a less unwieldy procedure could well include direct quality values in the form of rock mass "strength" estimates (12).

The inclusion of the hole size as a geometry parameter is a basic flaw in the relationship as geometry cannot be considered as part of the rock mass resistance to breakage. But the "rock" powder factor logic is valid.

The "rock" powder factor can be defined as a rock mass property similar to a "rock constant" in the Swedish Method.

The next logical step is to estimate the "design" powder factor for a specific blast.

A "design" powder factor will be a function of the following parameters:

- "rock" powder factor; the measure of the rock resistance to breakage by blasting.

- explosive energy; at present the "weight strength" is the conventional measure of the available energy. The effective use of this available energy depends on the actual detonation pressure achieved in the hole.

- geometry; effects of burden, spacing, charge height, blast length, hole deviation and availability of free face which describes the confinement of a charge and the difficulty in breaking out the burden.

Design powder factor = f(A. rock powder factors)
(B. explosive energy)
(C. geometry)

A further parameter which may account for some of the variation seen in underground blasting is stress. A high stress across the burden may inhibit loosening as it clamps the blocks in place. On the other hand, if the stresses are extremely high, then the rock may be overstressed and broken and consequently easier to displace. There are no references to this subject and it must form part of the basic research for this project.

The constants A, B and C can be reduced to a single site specific constant. The relationship is a simplification which gives an initial best estimate and would be calibrated as blasting proceeds. But it takes into account all the factors which influence the success of a blast. The other models, apparently, do not even take into account all these "factors" and they still involve a calibration phase.

2.5 Conclusions

Two very different modelling approaches can be followed; the theoretical or the empirical. Empirical includes an element of calibration but the theoretical approach, in practice, has also required considerable calibration. The theoretical approach can only describe the breakage process in a homogeneous rock under ideal detonating conditions. The rock is extremely inhomogeneous and the detonation can vary considerably. It can be argued that the theoretical approach is so unrealistic that it can be no better a predictor than the empirical method without calibration. The objective of this project is to produce practical and robust software and it is concluded that the empirical approach offers the most advantages. On this basis a modified Swedish Method should suffice so long as it includes a better estimate of the "rock" powder factor. But this does not include a design specific to crater blasting.

Crater blasting, in reality, is no different from blasting with long cylindrical holes. The explosive detonates and attempts to displace the burden. The resistance to the displacement is provided by the rock mass. The major difference is that the blast is much more confined and a different set of constants will apply. But there should be no reason to use the complicated and basically unrealistic procedure of developing crater test curves as a "rock" powder factor should suffice.

3.0 FRAGMENTATION

The previous section discussed breakage of the rock mass and displacement of the broken rock into a muckpile. The calculation of the energy required to loosen the rock mass is normally fairly straightforward. The prediction of the resulting muckpile is far from straightforward.

Firstly, there is considerable debate about the growth and extent of fresh fracture and secondly, what part of the detonation cycle contributes to which form of fracture; is it shock or gas pressure that is primarily responsible for fracture? (13).

The material which is blasted is already flawed, as discussed in the previous sections and illustrated in Figures 1, 2 and 3, and the resulting muckpile will contain the following material (Figure 13):

- Very fine material from the crushed zone within 1 to 5 radii of the blasthole.
- Fine to coarse material which is surrounded by fresh surfaces which mostly originates from within intact blocks traversed by a charged portion of the hole.
- Coarse to oversize material which is surrounded by a combination of fresh and existing surfaces which

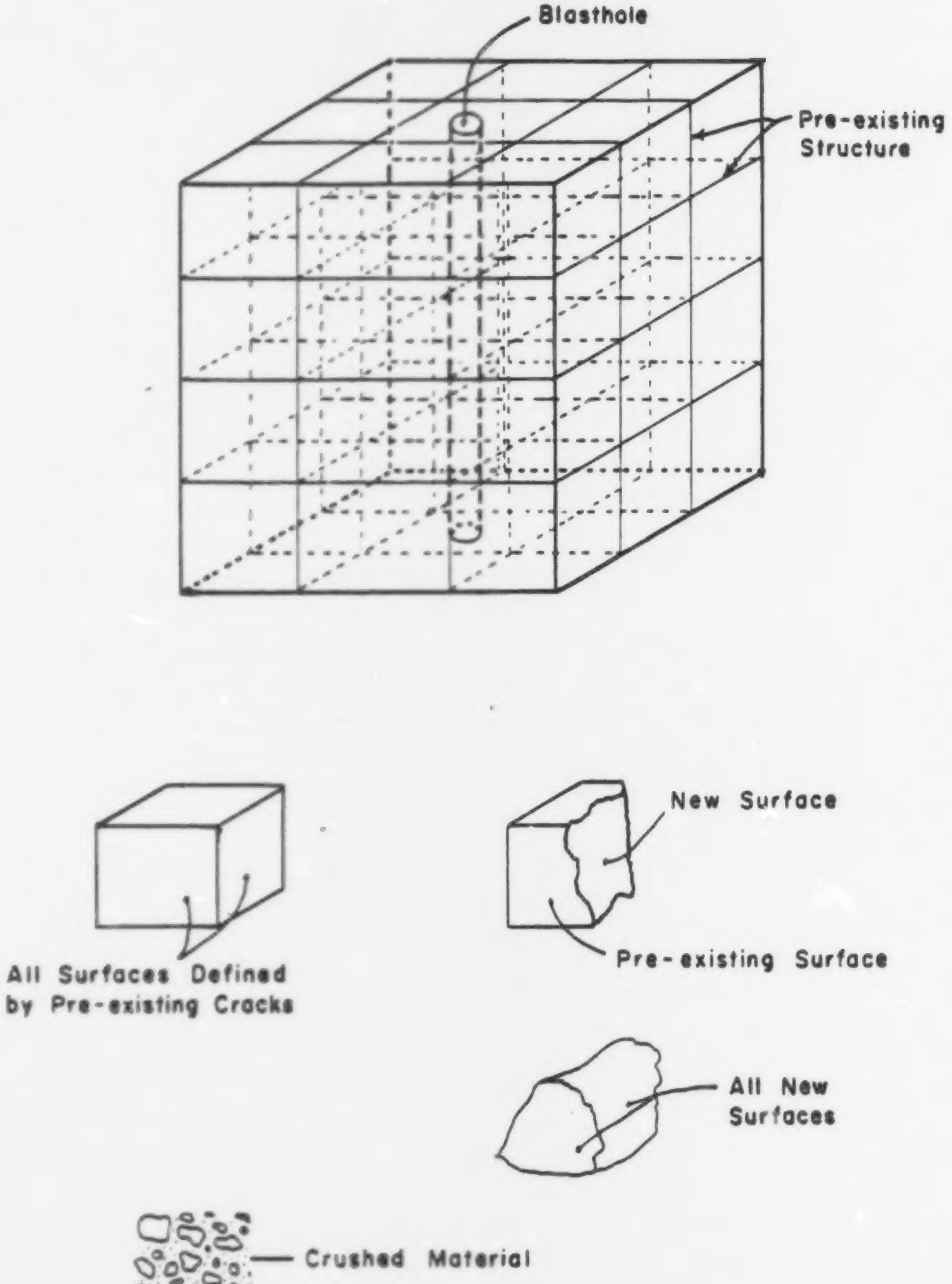

Blasthole
Pre-existing Structure
All Surfaces Defined by Pre-existing Cracks
New Surface
Pre-existing Surface
All New Surfaces
Crushed Material

FIGURE 13. INHERENT FRAGMENTATION

originates from large blocks intersected by charged hole, from interblock impact and from tearing and bending during burden movement.

- Coarse to oversize material which is surrounded by existing surfaces.

The proportion of each type of rock block in the muckpile will depend on a number of factors:

- explosive energy and partition into shock and heave.
- rock resistance to crack formation.
- geometry and timing of the blast.
- frequency of existing rock cracks.
- rock mass competence.

A completely homogeneous unflawed material will result in a muckpile which contains only blocks bounded by fresh cracks. Alternatively a very loose incompetent rock mass with frequent pre-existing fractures will result in a muckpile in which nearly all the material is bounded by pre-existing surfaces. In the second case it is unlikely that the amount of explosive alone will make much difference to the fragmentation as fragmentation can only be improved by intersecting more blocks with the charged portion of the hole. Which implies a decrease in the hole size to maintain the powder factor whilst decreasing the pattern.

There is considerable argument concerning the number of fresh cracks originating from a blasthole which will cross pre-existing cracks rather than follow those cracks. The influences can be illustrated figure 14. There is a strong body of opinion that concludes that fresh cracks will terminate at pre-existing joints. But it seems logical to assume that very tight irregular joints may not arrest crack growth. A detailed study of cratering experiments (7) concluded that radial fractures are "captured" by the first pre-existing fracture encountered. The authors go further ....

> "The use of cratering experiments to a free face to evaluate the effect on particle size distribution by explosive type, explosive geometry, or depth of burial of the charge appears to be futile. Only the breakage in the immediate vicinity of the charge changes and most of the rock is broken along pre-existing flaws. Particle size distributions for simple cratering experiments using charge weights from a few grams to 100 kg appear to be almost totally controlled by the pre-shot geology, that is, the distribution of flaws, joints and fractures......"

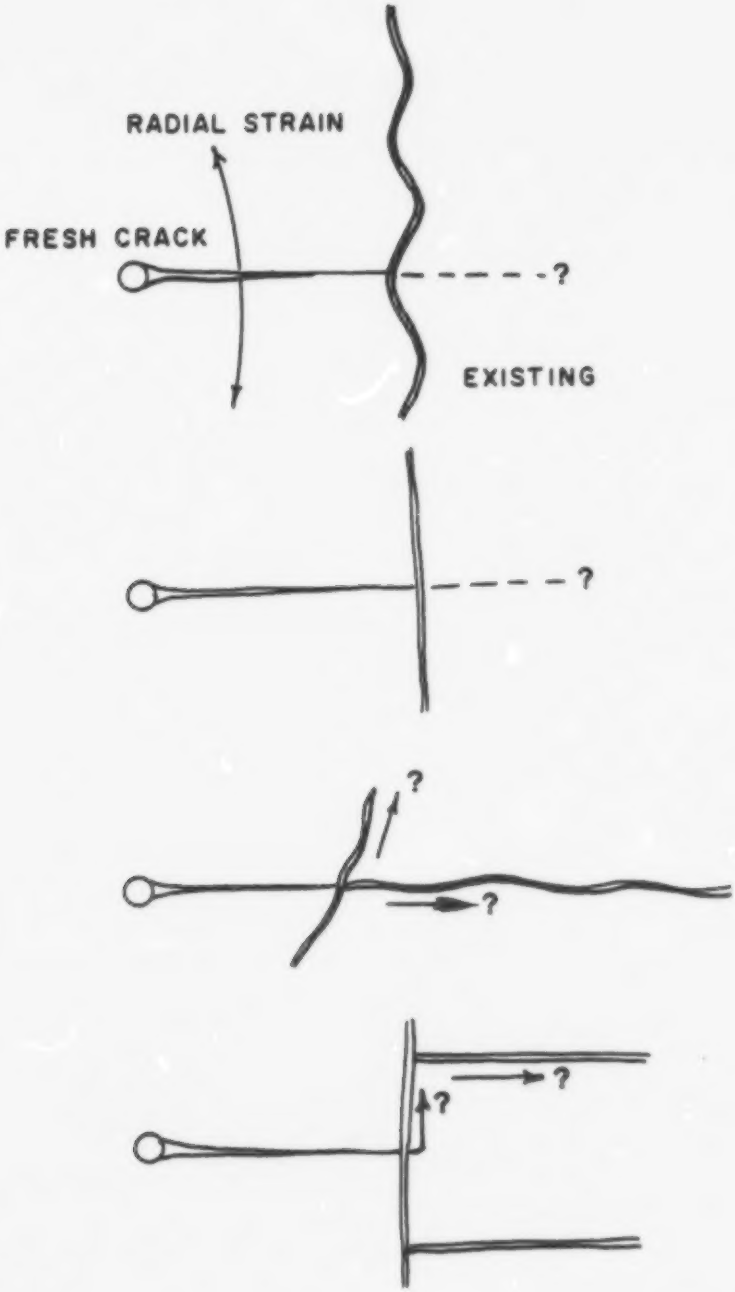

RADIAL STRAIN
FRESH CRACK
?
EXISTING
?
?
?
?
?

FIGURE 14. CRACK PATHS AND EXTENSION PAST EXISTING CRACKS DEPENDS ON JOINT PROPERTIES.

Although the influence of the structure is recognized there has been considerable research into fragmentation of homongeneous material and the development of relationships which estimate fragmentation but do not include terms for the structure. This will be illustrated in the following discussion.

Models which can be used for the evaluation of fragmentation fall into three categories:

- Strain models; as discussed in section 2.2.

- Empirical models; these are usually statistically derived relationships which have originated from estimations of size distributions from crushing and grinding processes.

- Rock fabric models; statistical representation or actual counting of discontinuities to estimate the existing size distribution of the blocks defined by the jointing plus an assessment of the additional fracture due to the blasting process.

3.1 Strain Model

A strain model can be developed from the relationships which are used to define the growth of cracks (8) as described in Section 2.2. and will not be repeated here. A crack growth pattern can be defined as illustrated in figure 15 (14).

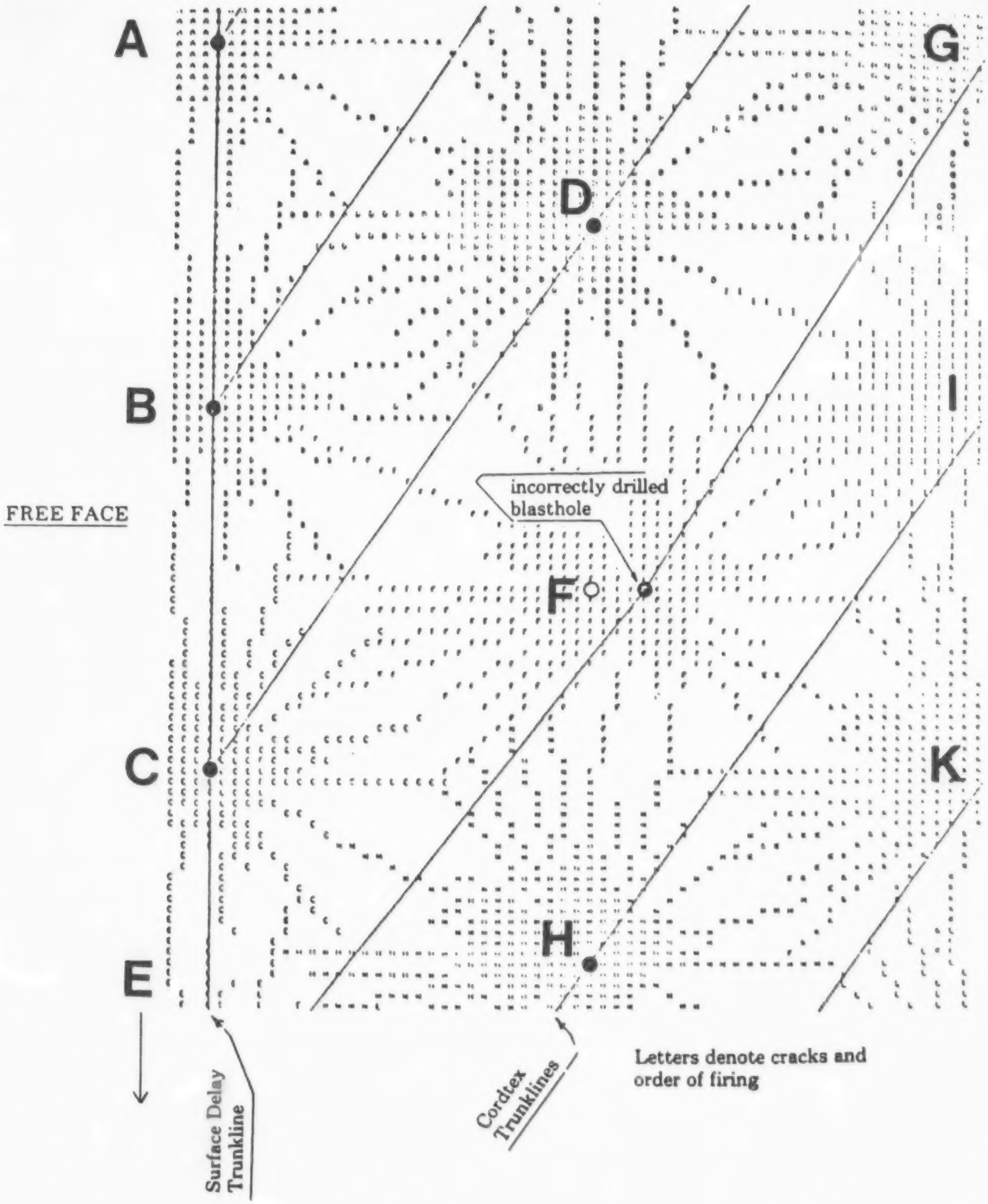

A
B
C
D
E
F
G
H
I
K
FREE FACE
incorrectly drilled blasthole
Surface Delay Trunkline
Cordtex Trunklines
Letters denote cracks and order of firing

FIGURE 15. EXAMPLE OF COMPUTER-GENERATED CRACK PATTERNS FOR FRAGMENTATION ANALYSIS (G. HARRIES).

This method has been extended (15) to include powder factor, hole pattern and timing. It relies on equations or rules to govern the effect of the variables in the system but with most of the algorithms originating from model scale blasting in perspex and marble. Figure 16 illustrates the geometrical parameters that are used. The general description of a radial crack pattern is as follows:

$$N_L = (L_{max}/L)^E$$

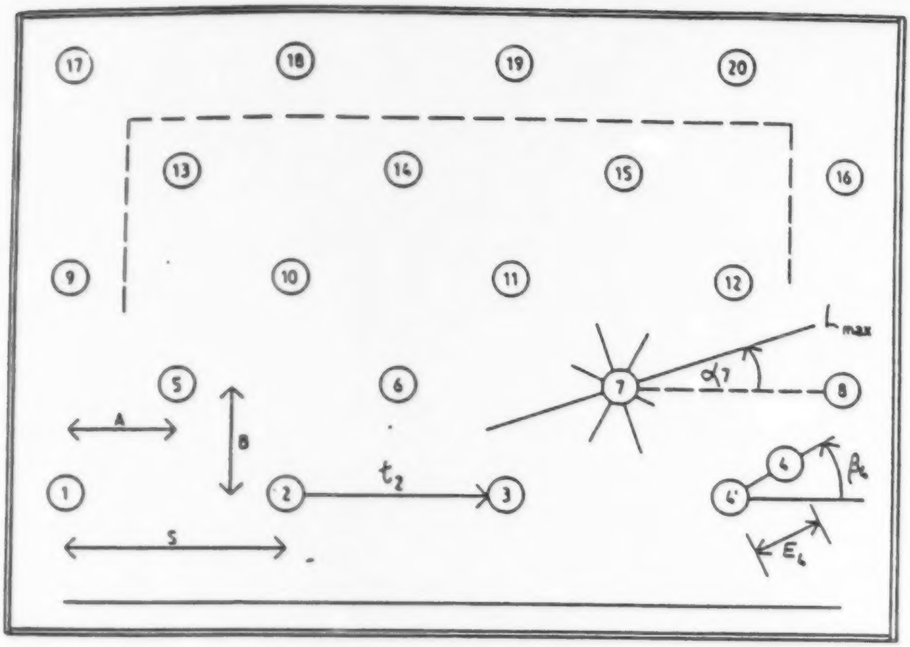
where L = crack length

N_L = the number of cracks of length L or longer

L_{max} = length of the longest crack

E = crack pattern exponent

Much of the confirmatory work was carried out in kimberlite. It was found that fragmentation with ANFO was adequately described by using 20 cracks per hole with $L_{max} = 68$ borehole diameters and $E = 1.5$. The longest crack was oriented parallel to the free face. Growth of cracks was constrained by 1.5 times the burden. The crack pattern from each hole was oriented at a different random angle to the free face. A number of other rules were developed to constrain the growth of cracks.


17
18
19
20
13
14
15
16
9
10
11
12
L_{max}
5
6
7
α_7
8
A
B
4
β_4
1
2
t_2
3
4'
S
E_4

Symbol	Shown at Hole	Meaning
B	2;5	Burden (as the holes are fired)
S	1;2	Spacing (as the holes are fired)
A	1;5	"Next-row offset"
t_2	2;3	Time delay between firing holes 2 and 3
E_4	4	Normally distributed error in hole position
β_4	4	Random angle $0 \leq \beta < 2\pi$
L_{max}	7	Length of longest crack
α_7	7	Random angle $0 \leq \alpha < 2\pi$
- - -	13;17	Boundary of Monte-Carlo survey

FIGURE 16. VARIABLES IN COMPUTER MODEL

The fragmentation in the model is in the form of wedges and it was assumed that the wedges were truncated by secondary fragmentation. The assumption that cracks are wedge shaped is supported by other work in homogeneous rock (16).

The method of assessing the fragmentation was to use a Monte-Carlo survey of a plane through the mass and the size of the biggest fragment was determined at each point selected. The method used is illustrated in Figure 17.1. It was assumed that each fragment modelled occurs with the same frequency in planes higher or lower in the block. A fragment size distribution was assembled from 250 fragments.

It is interesting to note that the model was used to compare fragmentation prediction from the Kuz-Ram relationship (17) which is discussed in Section 3.2. The agreement was very good and was taken as supplying further support to the hypothesis (14) that fragmentation by blasting is largely determined by radial fracture. This is not really surprising in a homogeneous material with no pre-existing cracks. The radial growth of the cracks with fewer longer cracks as radial distance increases will give the typical size distribution that is characteristic of any rock block distribution. The large scale tests were in kimberlite which is a significantly less jointed rock than is usually encountered. Further, the model assumptions were adjusted until a reasonable fit was achieved. The Kuz-Ram relationship also ignores the presence

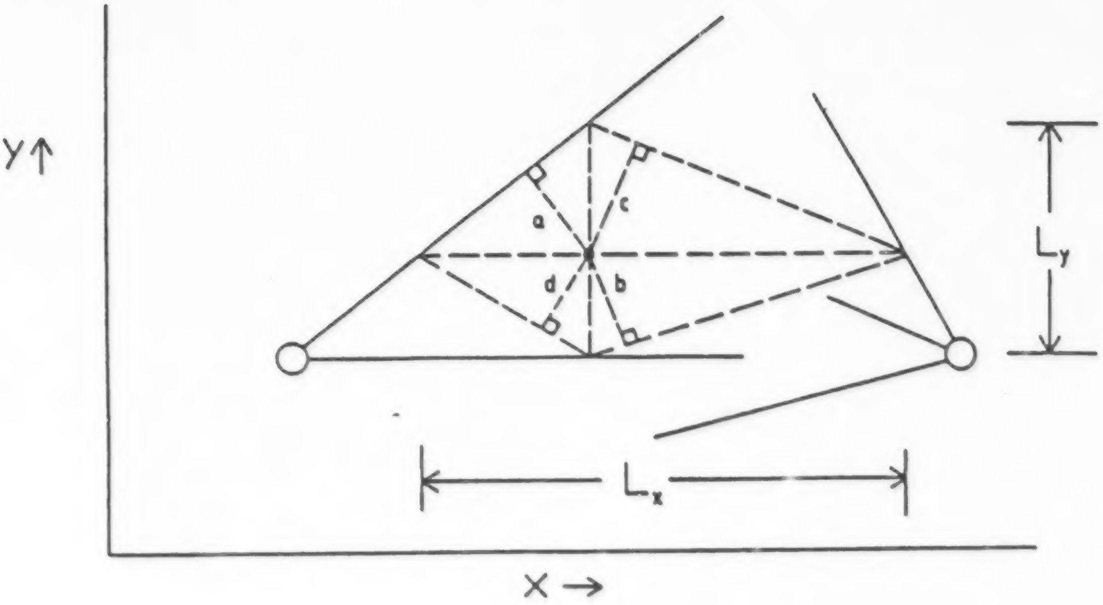

Y↑
a
c
d
b
L_y
L_x
X →

FIGURE 17.1 DERIVATION OF FRAGMENT SIZE BETWEEN RADIAL CRACKS. SIZE IS SHORTEST OF L_x, L_y, a+b, c+d.

of pre-existing cracks. Nevertheless, the procedures described are realistic and useful so long as pre-existing cracks can be included.

An example of a similar strain model (18) has been described based on the pressure pulse developed by the detonating explosive and the resulting radial and tensile stresses. Peak tensile stresses were statistically correlated to fragmented particle size distribution, as illustrated in Figure 17.2, to estimate fragmentation. The higher the tensile stress the finer the fragments.

Both methods assume that the rock is elastic and use as input those properties which control the elastic stress wave propagation. They do not include any parameters which describe the rock mass and the pre-existing cracks.

A larger charge or more powerful explosive will result in more crushing and fracturing but this effect may well be confined to areas close to the borehole. The coarse fragmentation may be largely unaffected unless the pattern is changed significantly as illustrated in figure 18.

3.2. Empirical Models

There has been a considerable amount of fragmentation study based on "comminution theory" and "work index" from the

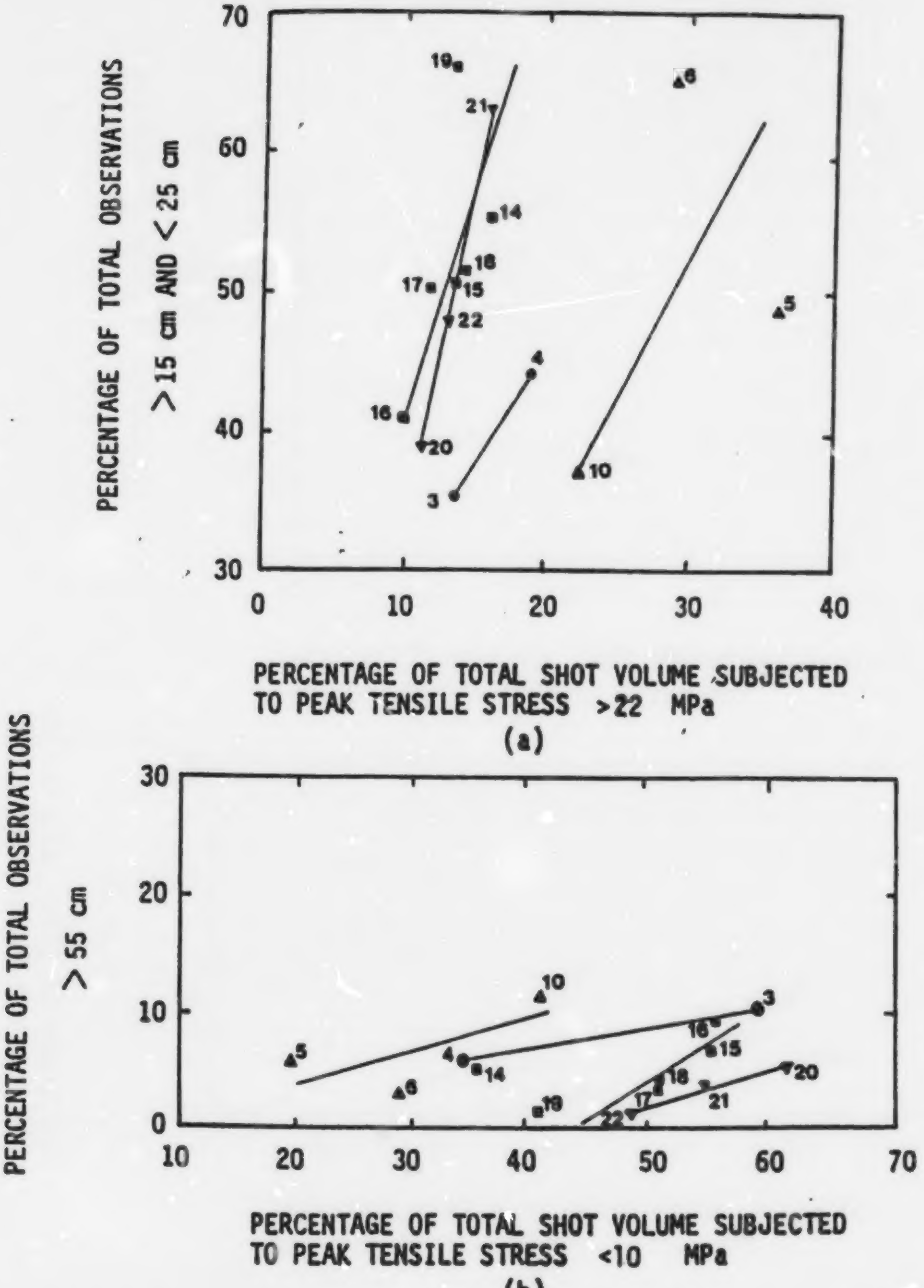
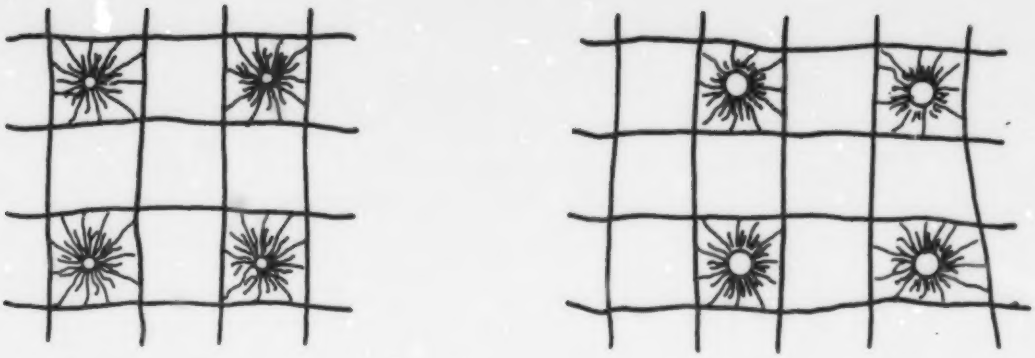
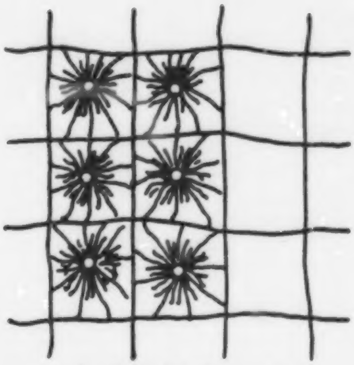


FIGURE 17.2 Percentage of Total Observations (a) >15 cm and <25 cm Versus Percentage of Shot Volume Subjected to Peak Tensile Stress >22 MPa and (b) >55 cm Versus Percentage of Total Shot Volume Subjected to Peak Tensile Stress <10 MPa.



(a) Small Holes

(b) Large Holes



(c) Small Spacing

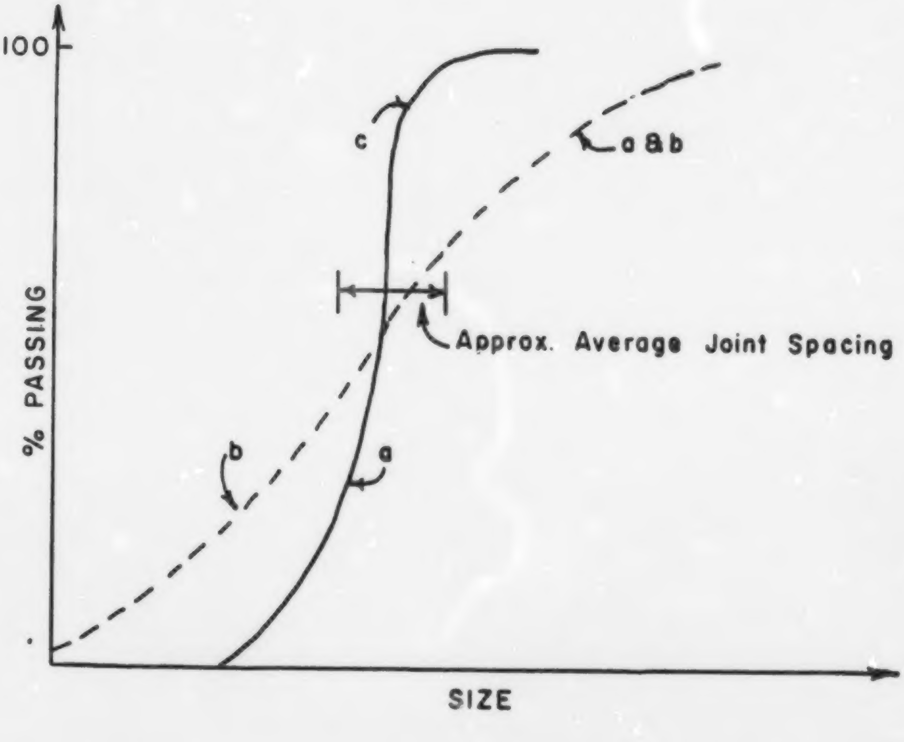


FIGURE 18. FRAGMENTATION MAY DEPEND MORE ON HOLE SPACING THAN POWDER FACTOR.

research on crushing and grinding. This route is difficult to reconcile with the basic differences in the action of breakage although the shape of the curve describing the size distribution may be similar.

The most frequently quoted work concerns that of Kuznetsov (17) which correlates the mean fragment size with the blast energy, hole size and rock strength. In addition the Rosin-Rammler curve (19) has been found to give a reasonable description of size distribution in blasted rock (20).

The Kuznetsov and Rosin-Rammler work has been used by Cunningham (21) in a rather elegant form to estimate the fragmentation resulting from blasting. The development of Cunningham's model is quoted below"......

Kuznetsov's equation is given by

$$\bar{x} = A\left(\frac{V_o}{Q}\right)^{0.8} Q^{1/6}$$

where $\bar{x}$ = mean fragment size, cm.

A = rock factor = 7 for medium rocks
10 for hard, highly fissured rocks
13 for hard, weakly fissured rocks

V_0 = rock volume (cubic metres) broken per blasthole, taken as Burden x Spacing x Bench Height.

Q = mass (Kg) of TNT which is equivalent in energy to that of the explosive charge in each blasthole.

If Q_e = mass per blasthole of explosive being used, and

E = Relative Weight Strength of this explosive, (ANFO = 100), then the Relative Weight Strength of TNT = 115.

$$Qe \times E = Q \times 115$$

$$Q = Qe \frac{E}{115}$$

Then $\bar{x}$ becomes:

$$\bar{x} = A\left(\frac{V_\circ}{Q_e}\right)^{0.8} Q_e^{1/6}\left(\frac{E}{115}\right)^{1/6} \left(\frac{E}{115}\right)^{-0.8}$$

which becomes

$$\bar{x} = A\left(\frac{V_\circ}{Q_e}\right) Q_e^{1/6}\left(\frac{E}{115}\right)^{-19/30}$$

The expression $\left(\frac{V_\circ}{Q_e}\right)$ is the inverse of the specific charge or powder factor, K, kg/cubicmetre.

Thus in the form most useful to explosives engineers the relationship becomes

$$\bar{x} = A(K)^{-0.8} Q_e^{1/6} \left(\frac{115}{E}\right)^{19/30}$$

This is readily inverted to obtain the necessary powder factor if a certain mean fragmentation is required.

Rosin-Rammler Equation

$$e^{-\left(\frac{\bar{x}}{xc}\right)^n}$$

The mean fragmentation is given the previous relationship;
$\bar{x}$ when R = 0.5

therefore $$e^{-\left(\frac{\bar{x}}{xc}\right)^n}$$

and $$x_c = \frac{\bar{x}}{(0.693)^{1/n}}$$

'n' determines the shape of the curve and commonly varies between 0.8 and 2.2. High values indicate uniform sizing, while low values result in higher proportions of both fines and oversize.

It would normally be desirable to have uniform fragmentation, avoiding both fines and oversize, so high values of 'n' are preferred.

Parameter	change in 'n'
Burden/Hole Diameter	decreases
Drilling accuracy	increases
Charge Length/Bench Height	increases
Spacing/Burden	increases
Rectangular changed to staggered patterns	increases

The algorithm for calculation of 'n' is:

$$n = (2.2 - 14\ B/d)\ (1 - W/B)(1 + (A - 1)/2)L/H$$

where B = (burden (m)

d = hole diameter (mm)

W = standard deviation of drilling accuracy (m)

A = spacing/burden ratio

L = charge length above grade level (m)

H = bench height (m)

If a staggered drilling pattern is employed, "n" as detailed above is increased by 10%.

A number of limitations are identified. One of these is most illuminating "..... the jointing and homogeneity of the ground require careful assessment as fragmentation is often built into the rock structure especially when loose jointing is more closely spaced than the drilling pattern...." (21) It is further remarked that in trials it was found that the rock factor was never lower than 8 or higher than 11. This implies that the rock mass can only exert a relatively small influence on the resulting fragmentation. This is a direct contradiction of the initial statement or an admission that the model only applies within a fairly narrow range of conditions.

Very limited calibration for Kuz-Ram has been published and much of this is with small model tests or in weaker rocks where the presence of pre-existing structure is less important. The "rock factor" does not include any reference to structure and is really an estimate of "blastability", as discussed in preceding sections, and can be compared with the "specific" charge. It assumes that if the charge is greater then the fragmentation will be improved and, therefore, fragmentation varies inversely with the degree of breaking difficulty which is not always the case.

But the model remains useful. It is very simple and makes realistic assumptions. It should be valid as a comparative tool so that the affect of changing different parameters within a single rock mass environment can be evaluated.

The criticism arises from the previous arguement of how many of the cracks which originate from the borehole actually cross pre-existing cracks.

Effect of Pre-existing Cracks

Lownds (15) believes that the fragmentation from the crack model can only be explained by allowing explosive induced cracks to cross pre-existing cracks, but this is possibly because he does not include any estimation of the effect of pre-existing cracks in the model. Harries (14), though, believes that most cracks tend to terminate at pre-existing cracks and is attempting to include the constraint in his crack model estimation. It was earlier quoted (7) that pre-existing cracks intercept most, if not all, of the primary cracks induced by the detonation.

Considerable model work has been carried out to investigate the influence of pre-existing cracks (22, 23). It has been concluded in the latter reference that ..... "When the charge concentration is varied in the different materials tested, the fragmentation k_{50} is changed most in the models with homogeneous material (5 times), a little less in models with weakness planes parallel to the blasted surface (3 times) and least in model material with parallel and perpendicular weakness planes to the blasted surface (2 times). The conclusion being that the more weakness planes, the more difficult to change the fragmentation..."

This is extremely important as it demonstrates that the specific charge and the fragmentation are not, necessarily, directly related. A knowledge of not only the "blastability" but also the "natural" fragmentation is necessary for an estimate of the fragmentation after blasting.

3.3 Fabric Models

One must conclude from the previous discussion that the rock mass structure has a fundamentally controlling influence on the fragmentation and that figure 18 is a reasonable summary of the problem. The previous models, both strain and the empirical Kuz-Ram, can predict, fairly satisfactorily, the fragmentation around the borehole and up to the first major pre-existing structural surface. Beyond this surface the fragmentation may be controlled by the pre-existing cracks.

It is the oversize which exerts the most influence on mining efficiencies and it would appear that once the hole pattern is greater than the average spacing of the joints then the joint spacing may determine the oversize (24).

There are a number of routes which can be taken in the estimation of the natural fragmentation:

- Calculation of "actual" block volumes from line surveys over the blast area.

- Estimation of the distribution of block volumes from a sampling of the joint data.

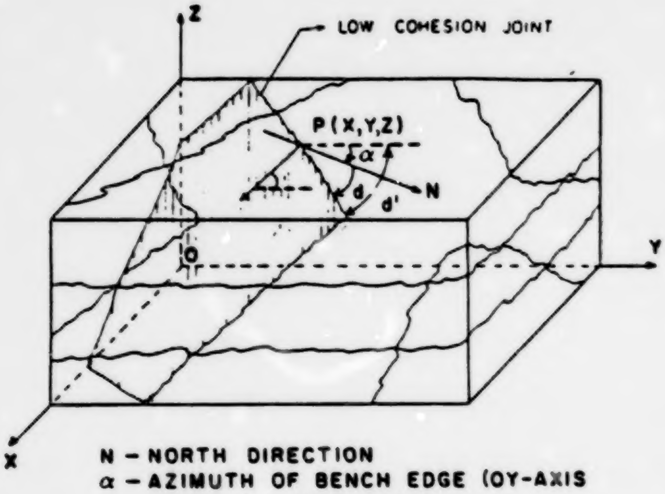
- Estimation of the grading curve from rock mass classification values.

These constitute varying degrees of difficulty and computational effort.

Calculation of Block Volumes

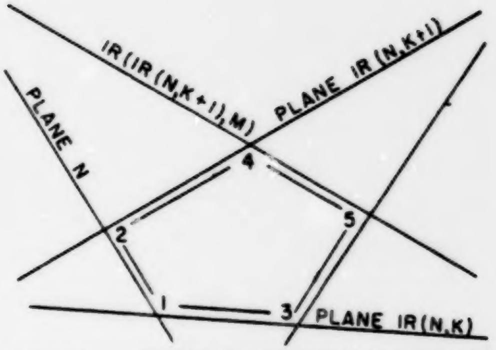
A method is described by Da Gama (25) in which joints are defined by their dip and strike parameters and are assumed to be continuous. Their position in space is determined figure 19a, and the intersection points calculated, fig. 19b, as a series of characteristic polygons. The volumes of elementary solids can then be calculated figure 19c. Actual joint data is used in the reference but data could be generated by randomly sampling the joint distributions determined from cell mapping.

A drawback of the system is the assumption that joints are continuous which will result in under estimation of oversize. There is very little description of the actual algorithms used and it is assumed that some approximation is implied in the reference. Considerable computational effort is required.

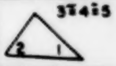
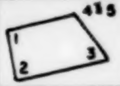
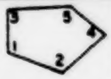

Z
LOW COHESION JOINT
P(X,Y,Z)
α
N
d
d'
O
Y
X

N - NORTH DIRECTION
α - AZIMUTH OF BENCH EDGE (OY-AXIS
d - STRIKE OF JOINT PLANE
i - DIP OF JOINT PLANE

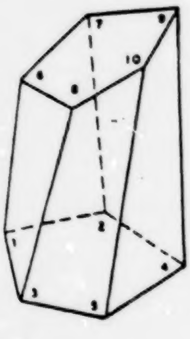
A) SCHEME OF A ROCK BENCH WITH JOINT PARAMETERS REQUIRED FOR THE STUDY.


IR(IR(N,K+1),M)
PLANE IR(N,K+1)
PLANE N
4
2
5
1
3
PLANE IR(N,K)

(K = 1,2, ... Number of segments in plane N)

No. of SIDES	FIGURE	CONDITIONS
3		IR(N,K) = IR(IR(N,K+1), M) or IR(N,K+1) = IR(IR(N,K), L)
4		IR(IR(N,K), L) = IR(IR(N,K+1), M)
5		IR(IR(N,K), L) ≠ IR(IR(N,K+1), M)

B) CRITERION FOR THE FORMATION POLYGONS.


7
9
10
6
8
1
2
4
3
5

TRIANGULAR PRISMS	TETRAHEDRA
1, 2, 4, 6, 7, 9	1, 7, 2, 4 1, 7, 4, 9 1, 6, 7, 9
1, 4, 3, 6, 9, 8	1, 6, 9, 3 1, 9, 4, 3 6, 9, 4, 8
3, 4, 5, 8, 9, 10	5, 10, 8, 9 8, 9, 3, 5 3, 4, 5, 8

C) CRITERION FOR THE DIVISION OF A BLOCK IN ELEMENTARY SOLIDS TO ALLOW VOLUME CALCULATION

FIGURE 19. ASSESSMENT OF "NATURAL" FRAGMENTATION.

Estimation of Blocks from Joint Frequency

An extremely elegant approach is described by Hudson (26). The spacing between joints along a line tends to a negative exponential distribution (Figure 20). This can be extended to the distribution of areas between joints intersecting a plane, which also shows little difference between random or orthogonal orientations (Figure 21). The argument logically extends to volumes deliniated by the intersection of joint planes.

All that is required to produce a distribution is the average spacing for each joint set. Hudson demonstrates that the error produced in using the average spacing as opposed to the actual spacing distribution is relatively small. "Relatively" is in the sense that uncertainty is inherent with design in rock masses.

From a distribution of block sizes it is easy to produce the grading curve which is required for a fragmentation assessment. The drawback of Da Gama's (25) procedures is accommodated in the mechanics of the line survey as discontinuous joints will not always be intersected, and therefore the process should not underestimate the oversize (Figure 22) (27).

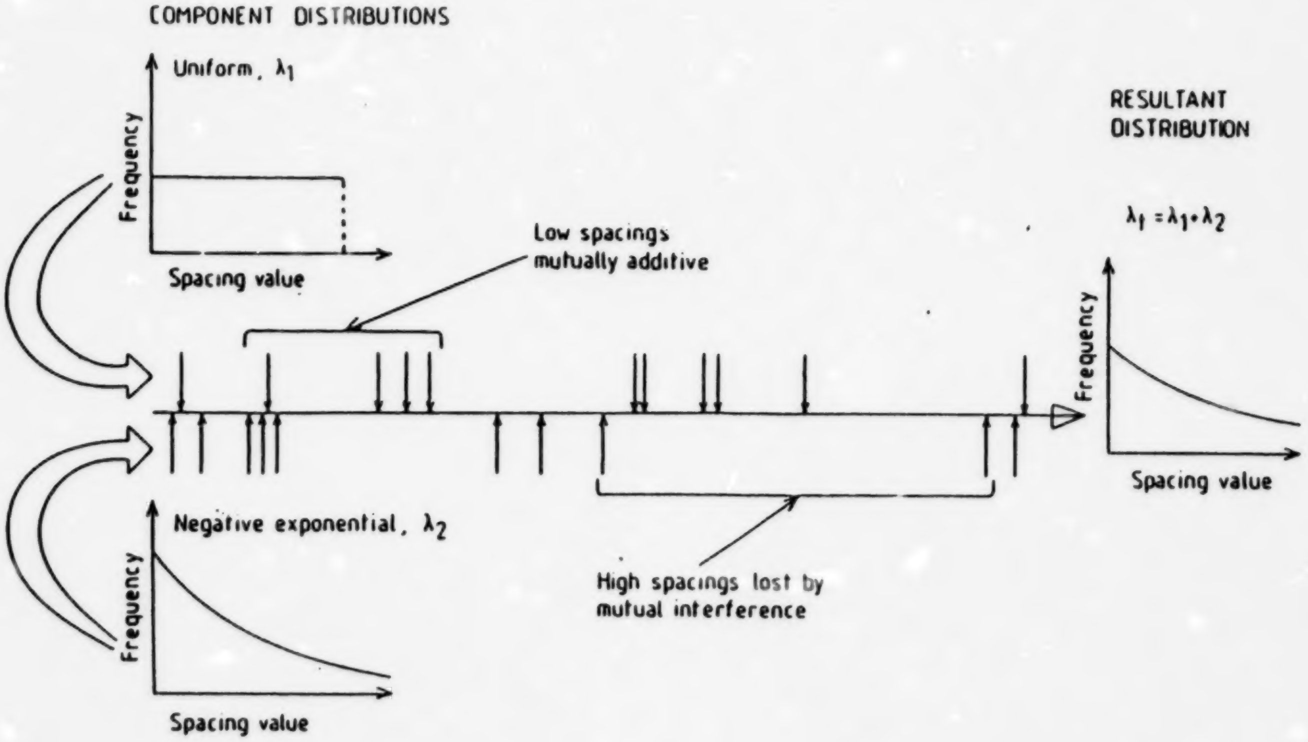

COMPONENT DISTRIBUTIONS
Uniform, λ_1
Frequency
Spacing value
Negative exponential, λ_2
Frequency
Spacing value
Low spacings mutually additive
High spacings lost by mutual interference
RESULTANT DISTRIBUTION
$\lambda_t = \lambda_1 + \lambda_2$
Frequency
Spacing value

FIGURE 20. COMBINATION OF TWO DISCONTINUITY SPACING DISTRIBUTIONS.

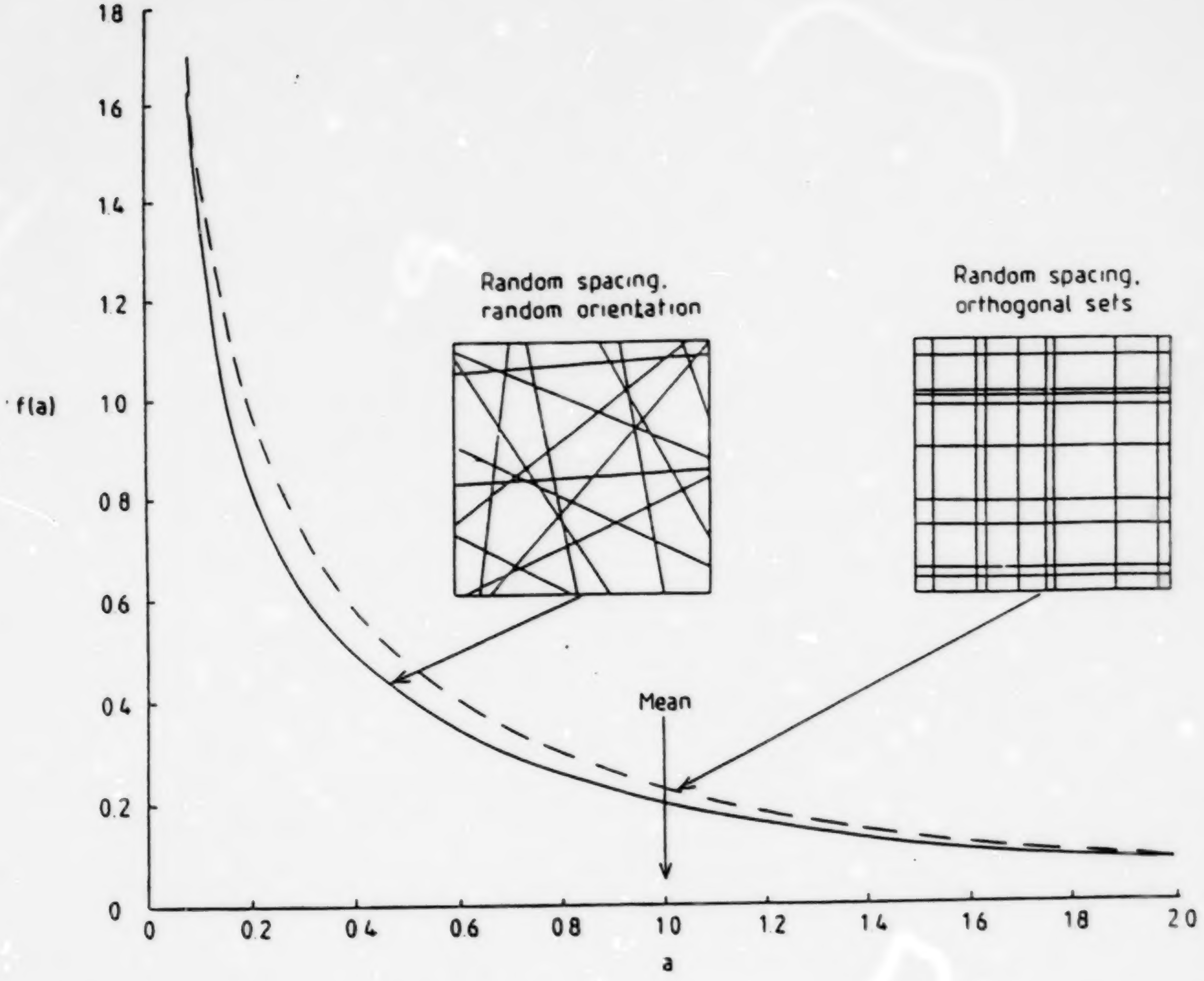
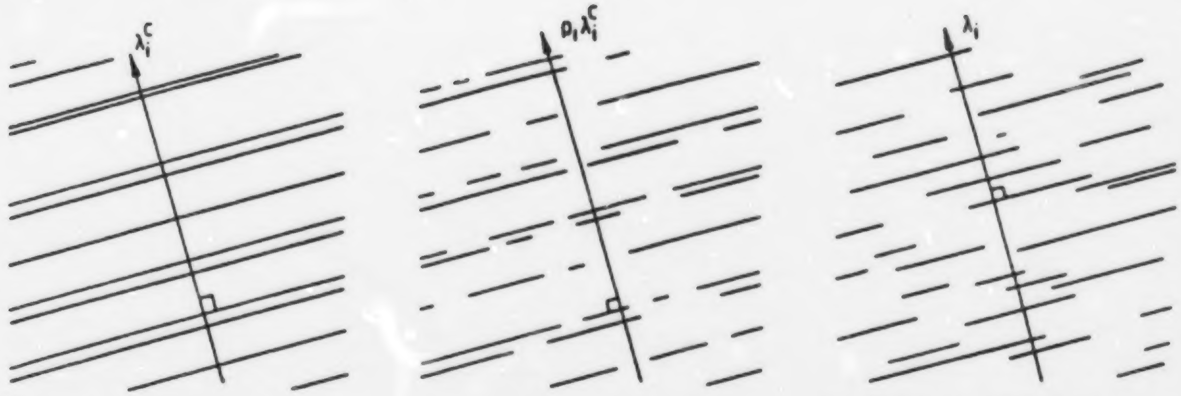


FIGURE 21. PROBABILITY DENSITY DISTRIBUTION OF AREAS FOR TWO TYPES OF RANDOM TESSELATION.



(a) Persistent discontinuities

(b) Impersistence produced by intermittent discontinuities

(a) Separate impersistent discontinuities

FIG. 22. TRACES PRODUCED BY SETS OF PERSISTENT AND IMPERSISTENT DISCONTINUITIES INTERSECTING A PLANE SURFACE.

(a) Persistent discontinuity planes.
(b) Intermittend discontinuity planes
(c) Separate impersistent discontinuity planes.

Estimation of Blocks from Classification Data

The application of rock mass classification procedures has had considerable success in the estimation of the cavability of the ore in block caving mines. Competent rock masses require much larger undercut areas to produce a cave than less competent rock masses. It has also been apparent that the less "cavable" a rock mass the greater the problem with oversize and secondary blasting. Laubscher (28) has produced a design method for determining draw point spacing, secondary blasting requirements and draw methods based on the RMR classification techniques (12) or the Fracture Frequency (Figure 23). This is an indirect estimation of the fragmentation as it estimates the effects of oversize. Oversize has a considerable effect on cost and therefore is the most important parameter in the assessment of fragmentation.

3.4 Conclusions

It is concluded that an estimate of fragmentation which does not include the jointing is fundamentally incorrect. The "crack" and Kuz-Ram models would appear to be limited to estimation of the fragmentation of blocks intersected by the charge. Analysis of the jointing can give an estimate of the natural fragmentation before blasting and a combination of this analysis and rock mass classification should give an estimate of the oversize after blasting.

CAVE MINING DATA RELATED TO RANGE IN PRIMARY
ROCK MASS CLASSIFICATION FRAGMENTATION

RMR CLASS and RATING	% OF ORE WORKED BY				SECONDARY BLASTING g/T	HANG UPS %	FRACTURE FREQUENCY/m
	GRIZZLY 14"	L.H.D. Cu. Yds.					
		3	5	8			
0							
	100						30
5		100	100	100	20	0	
							20
20	90						14
21	80	100	100	100			
4		80	85	90	80	15	5
40	40	60	70	80			2.5
41		40	60	70			
3	20	30	45	60	150	25/30	
							1.0
60		20	30	50			0.7
61		10	15	40			
							0.4
2	0	5	10	20	250	40/50	
							0.2
80		0	5	10			0.1

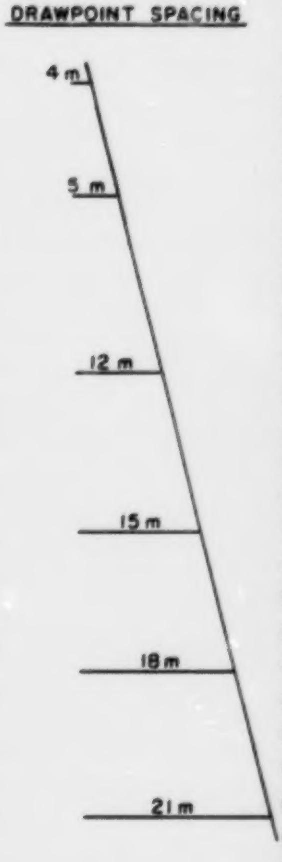


FIGURE 23. BLOCK CAVE DESIGN WITH FRAGMENTATION.

Since the "crack" and Kuz-Ram give similar results it is logical for this project to use the Kuz-Ram model.

If all the blocks are intersected by blast holes then Kuz-Ram model should give a good estimate of fragmentation. Once the hole spacing is greater than the average fracture spacing then the oversize will tend to be defined by the pre-existing fractures. An estimate of the coarse portion of the distribution will be given by the fracture spacing and will be similar to the natural fragmentation with some adjustment for bending, tearing and impact.

Work carried out at Mt. Isa (29) identified certain components in the mining system which are sensitive to fragmentation. Their proportional influence on the overall potential cost reduction is shown below:

-	secondary breakage	25%
-	LHD operations	40%
-	chute and tipple maintenance	3%
-	rail haulage	15%
-	primary crushing	5%
-	recovery losses and/or dilution	12%

Most of these cost reductions are directly related to the effects of oversize. It is interesting to note that no correlation between fragment size and LHD loading rate was indicated until a "boulder" was encountered, at which point the cycle time increased significantly.

It is evident that for mining purposes the estimate of "oversize" is the most important value and therefore the natural fragmentation estimate together with the drill hole spacing will be determining factors as shown in figure 24.

The final consideration will be the proportion of the rock adjacent to the explosive charge. Rock not adjacent to a charge will tend to follow the fragmentation curve estimated for the natural fragmentation.

4.0 DAMAGE

The previous two sections dealt with rock breakage and the resulting fragmentation. "Damage" can be described as the rock mass that is broken but which is not displaced into a muckpile by the expanding gases. Exactly the same factors which affect or control the breakage process will also affect the damage to the rock mass forming the resulting excavation. This is one definition of damage. The second is the affect of a detonation on existing excavations and installations.

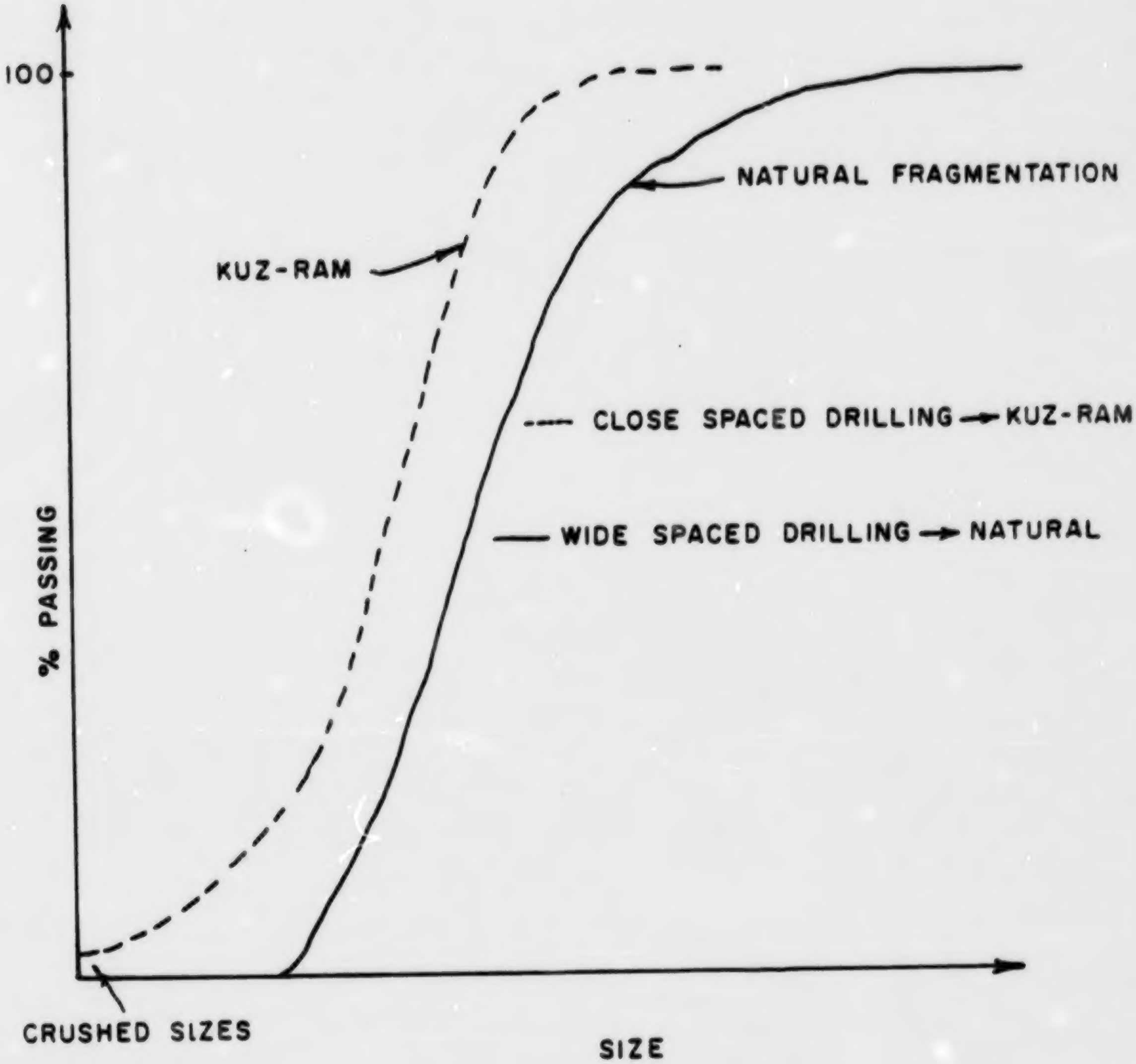


FIGURE 24. SIZE DISTRIBUTION BOUNDARIES

This project is concerned mainly with the design of blasts for open stoping methods. In this context critical "damage" is mainly confined to the open stope itself, as follows:

- Reduction in rock mass quality; a smaller excavation has to be used because of the damage to the rock mass.
- Dilution; the blast overbreaks the walls of the stope.
- Poor fragmentation; ore behind the blast is so badly damaged that it falls but is not properly fragmented.

The three definitions are essentially the same. The difference relates to the degree of damage. The description of rock breakage in Section 2.0 is also valid for the damage. The breakage process creates fresh cracks, extends and opens existing cracks and causes movement on joint surfaces, the result being that the rock remaining in the walls of an excavation has a reduced competence.

In the context of this project "damage" will be a value which is used in the stope stability assessment and is therefore a negative factor in the rock mass classification process.

Damage assessment has conventionally followed two routes. The rupture or strain model which is used in breakage assessment can also be used to define a rupture envelope both in front of a hole and behind that hole (Figure 9). This is evidently the procedure used in some of the proprietary software. The rupture envelope can be used to evaluate overbreak.

The second route is to measure or estimate the intensity and character of the seismic waves produced by the detonation. Both the intensity and character vary with the rock and distance from the detonation and also vary depending on the size and type of blast (30, 31). Typical examples from these two references are shown in Figure 25. The complexity of the wave form and the influence of the different wave properties, acceleration, velocity, amplitude and frequency, on the type and magnitude of the damage make seismic vibration an extremely complex subject.

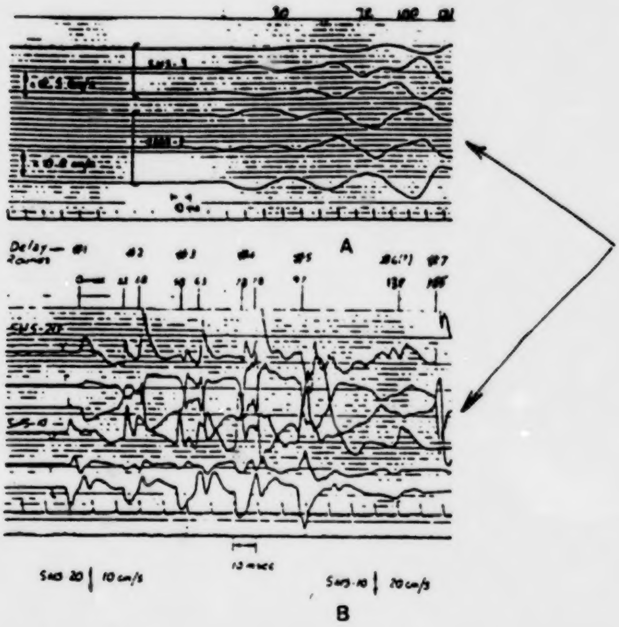
The approximation which is commonly adopted is to measure or estimate only the peak particle velocity PPV:

$$PPV = A\,2\pi f \quad (32)$$

where A = amptitude

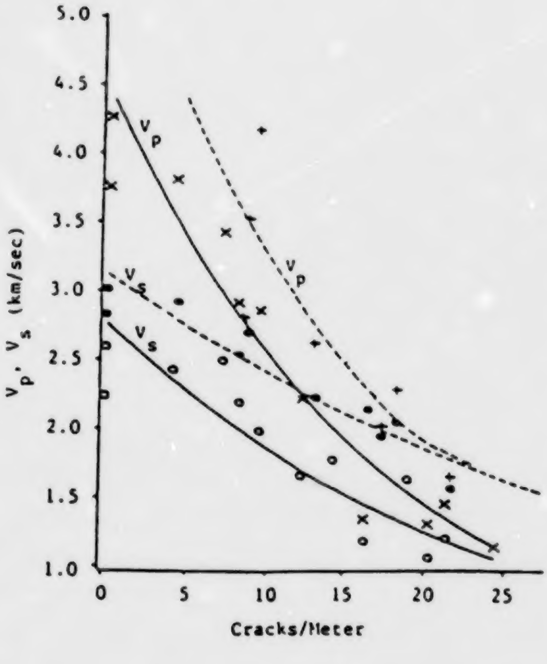
f = frequency

There are a considerable number of commercial instruments available which measure PPV and there is a body of data available for different blast sizes and types in varying rock mass environments.



TWO VERY SIMILAR BLASTS MONITORED FROM THE SAME DISTANCE BUT IN VERY DIFFERENT ROCK MASSES.

(a) Blasting Vibrations from an open pit coal mine (A) and an open pit metalliferous mine (B).



(b) Influence of cracks on V_p and V_s for different V_p/V_s ratio rock samples. ($\overset{+}{x}$) high ratio; ($\dot{o}$) low ratio.

FIGURE 25. INFLUENCE OF ROCK MASS ON SEISMIC WAVES.

4.1 Strain Model

The strain model has been adequately described in section 2.2. The number of cracks or the maximum length of a crack are values which could be used to define the extent or intensity of damage. Harries (8) assumes that ...."the distance to which blasting can affect the stability of a slope (open pit) is the distance to which a blast can produce one crack or extend an existing crack." Of course, a single crack may not significantly affect the stability but the generation of cracks can be taken as a potential for disturbance.

The strain method links explosive energy with elastic intact rock properties but it does not take into account the pre-existing damage in the form of joints and bedding planes. There is strong evidence that the crack process is arrested, or at least significantly altered, by the existing structure (7).

The intact elastic properties of a rock mass probably have the least influence on stability and it is the rock mass competence and strength which are most important. Likewise it is likely that the rock mass competence and strength dominate the rock mass resistance to damage, as argued in section 2.4. This does not alter the form and logic of the strain relationships and they may become more applicable if "tensile breaking strain" were replaced by a measure of the rock mass strength and N, the number of cracks, could be termed the "damage" number; applicable to the relationships in section 2.2.

4.2. Peak Particle Velocity Model

Peak particle velocity measurements can only be taken beyond the permanently damaged inelastic zone due to the intense damage. This is beyond the area where gases have any disturbing affect. Very close to the blast the PPV must be estimated by extrapolation.

It is shock and gas pressure which results in most of the permanent damage to a rock mass, not the seismic waves. But as shock and gas pressure produce the seismic wave then a value for the PPV must reflect the energy source which was the shock generated by the explosive.

Peak particle velocity can be estimated from the following relationship:

$$V = \frac{k\ W^{a}}{R^{b}}$$

where V = peak particle velocity, mm/s

W = charge weight per delay, kg

R = distance from blast, m

K, a, b = constants which may vary according to the structural and elastic properties at the blasting site.

There is a very wide spread of values for these constants which could be due to different explosives, geometries and patterns or to different rock mass properties.

In the absence of actual site data a reasonable first estimate of the vibration levels can be made by assuming the following average constraints:

$k = 700$
$a = 0.8$
$b = 1.6$

The above relationship is only reliable at fairly large distances from the blast, distances which are usually beyond the zone of interest of the mining engineer. He is usually interested in dilution, which may be within 3 to 15 m, or collapse of the excavation, again very close to the source of the disturbance, whilst the hole length may be 10 m long or longer. Close to a blasthole the rock does not register the combined damage potential from the whole charge but only a portion of that charge. Therefore the above equation is far too conservative. This problem can be resolved (33) by integrating the relationship to give:

$$V = k\,L^{a}\left[\int_{0}^{H} \frac{dx}{\left[R^{2} + (R\tan\Theta - x)^{2}\right]^{b/2a}}\right]^{a}$$

where L = charge mass per metre, kg/m

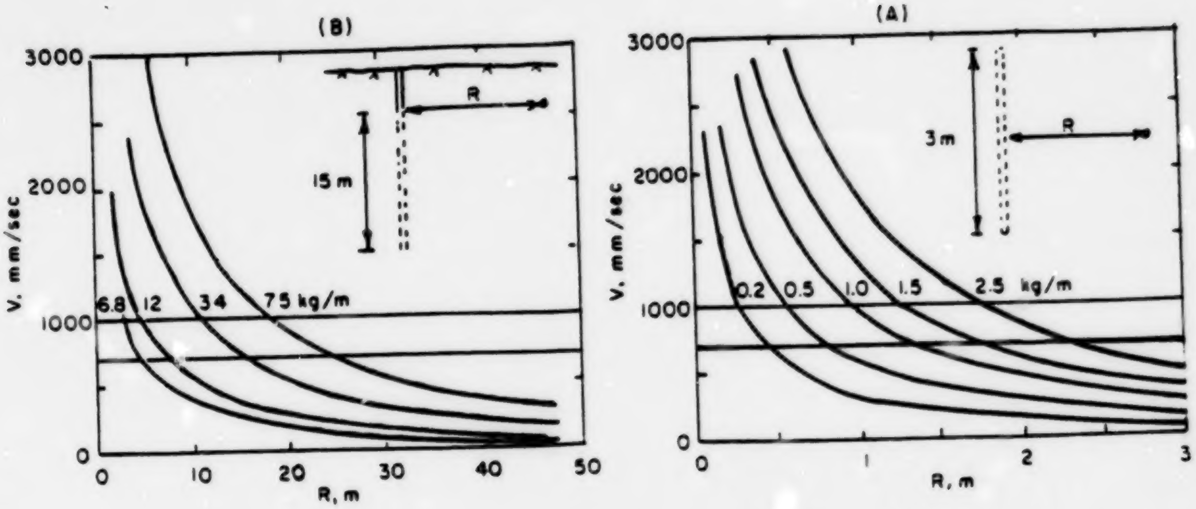
H = charge length, m

The results are shown in Figure 26 (a) with the reasoning behind the integration shown in Figure 26 (b).

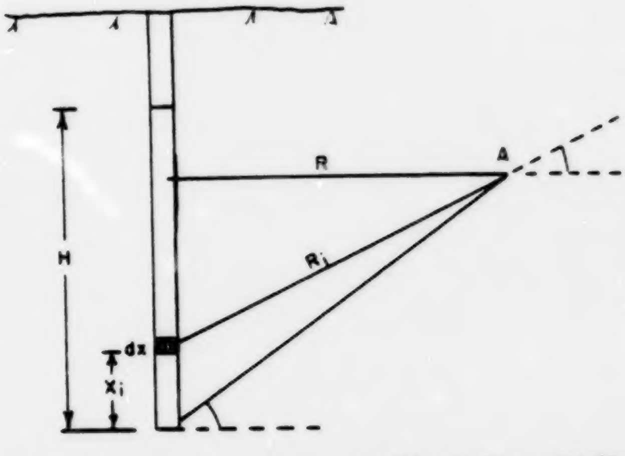
But this is only part of the solution. All that the relationship contains is the weight of explosive, distance and three site specific constants. Considerable effort is expended in obtaining site specific constants using a certain blast pattern, timing and explosive type. If, some time in the future, the hole size, pattern, timing or the rock mass in which blasting takes place are changed, a completely different vibration relationship may result. The spread of data which can result is shown in Figure 27 (34) which suggests a 25 fold difference in results dependent on the following:

- rock mass characteristics
- explosive power (as distinct from charge mass)
- confinement
- coupling
- timing

Within a single operating environment the spread of data would reduce but could be as much as five fold. It is therefore important to identify the parameters which contribute to this variation.



(a) ESTIMATE PEAK PARTICLE VELOCITY (AFTER HOLMGREN, REF. 18)



Integration over charge length to calculate particle velocity at an arbitrary observation

(b) INTEGRATION OF STANDARD VIBRATION RELATIONSHIP CLOSE TO CHARGE (AFTER HOLMGREN, REF. 18).

FIGURE 26. ESTIMATE OF PEAK PARTICLE VELOCITY CLOSE TO A CHARGE.

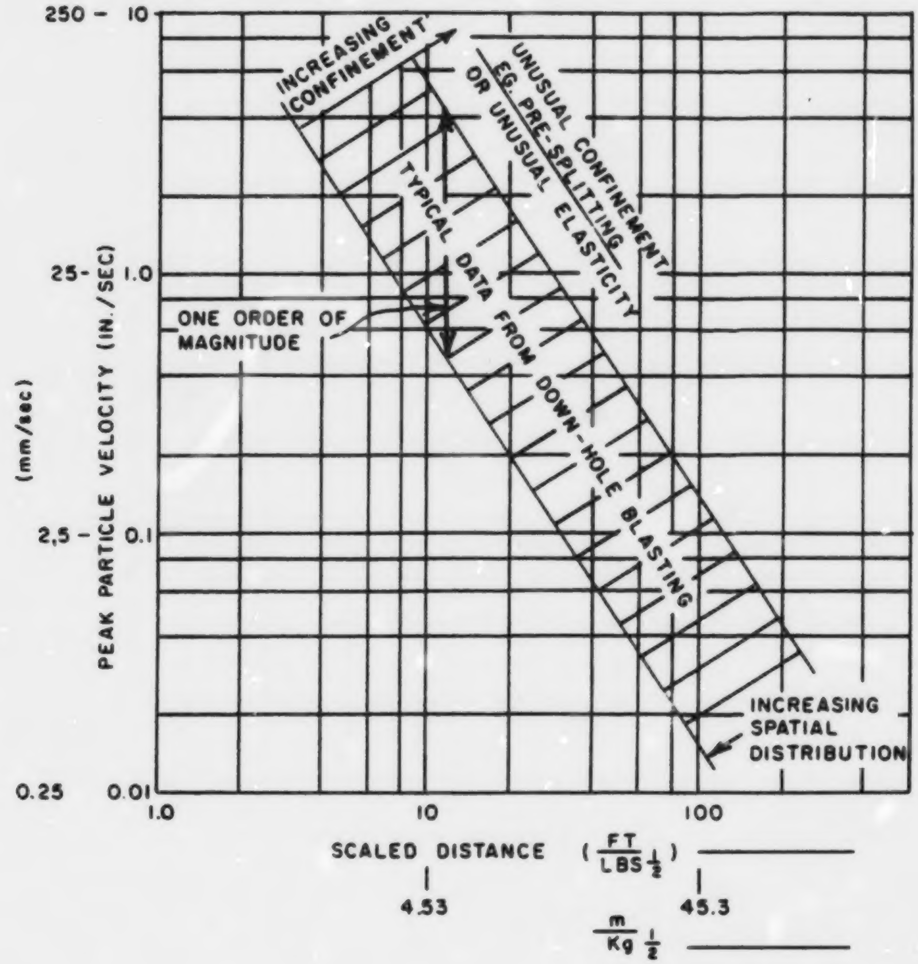


FIGURE 27. VARIATION IN PEAK PARTICLE VELOCITY (AFTER ORIARD, REF. 19)

4.2.1. Controllable Influences

The reaction of an engineer to the considerable variation in PPV from one blast to another may be to increase the constraints placed on the blasting to very conservative levels. Uncertainty can be reduced by detailed and continued measurement but measurement is after the fact. Damage is also "after the fact" and overbreak cannot be put back. The factors which contribute to the variation in PPV have been identified and estimates of their affect can be made.

Explosive Power

The relative weight strength of explosives, or other measures of energy, can vary considerably from one explosive to another. ANFO with a weight strength of 100 can be compared to an aluminized slurry with a weight strength of 160, for example.

The most widely used method for comparing relative weight strengths is:

$$S(\text{rel. wt. str.}) = {}^{5}/_{6}\,\frac{Q}{Q_0} + {}^{1}/_{6}\,\frac{V}{V_0} \quad \text{(ref. 35,36,37)}$$

where Q_0 and Q = heat of explosion for the reference explosive and the new explosive respectively.

V_0 and V = gas volume released.

The weight strength concept describes the magnitude of the expansion work the explosive can perform during a blast. But not all this energy is "useful" and the amount will depend on the explosive chemical constituents and the characteristics of the rock mass. It is stated (36) that rock breakage and primary fragmentation are completed when the detonation products have expanded to a volume of about ten times the borehole volume, as illustrated in figure 28. Damage beyond the zone of primary fragmentation is due to the explosive gases propagating into the existing cracks in the rock mass.

Some authorities (37) suggest that the amount of rock broken is not directly proportional to the relative weight strength and use the following relationship:

$$\text{New volume broken} = \text{Old volume broken} \times \left(\frac{\text{relative energy new explosive}}{\text{relative energy old explosive}}\right)^{2/3}$$

The available energy in the explosive can be estimated but the source of damage is in reality the borehole pressure. This will depend on how tightly it is charged and is a function of the coupling ratio.

Coupling

The difference between the charge diameter and hole diameter can exert the single most important influence on the borehole pressure.

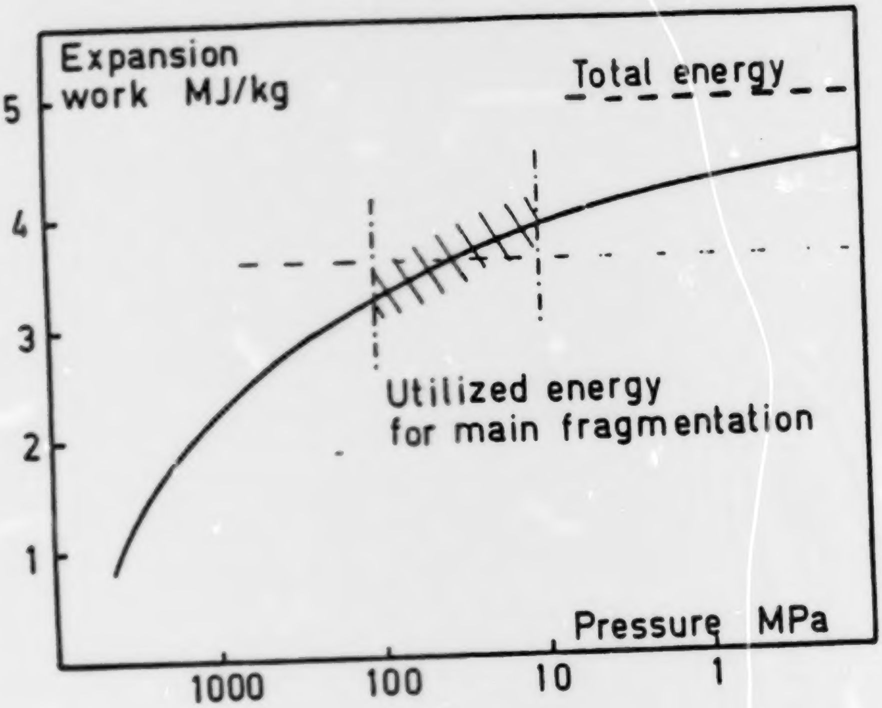


FIGURE 28. EXAMPLE OF THE EXPANSION WORK AS A FUNCTION OF THE PRESSURE IN THE REACTION PRODJCTS FOR AN ALUMINIZED WATER GEL EXPLOSIVE.

The effective borehole pressure can be calculated as follows:

$$Pb(dc) = Pb(c) \times (CR)^{2.6} \quad (38)$$

Where Pb(dc) = borehole pressure for a de-coupled and/or decked charge, MPa

Pb(c) = borehole pressure for the fully coupled explosive (obtainable from Explosive Supplier)

CR = coupling ratio, which is the ratio of the diameter of the charge to the diameter of the hole.

It should be noted that a 10% reduction in charge diameter from fully coupled will reduce the borehole pressure by approximately 24%. When using cartridges, as opposed to poured or pumped explosive, a coupling ratio of less than 0.9 is typical.

It can be argued that the potential to do damage and to generate vibration varies directly with the borehole pressure and can be accounted for by reducing the charge weight in proportion to the ratio of the theoretical borehole pressure to the actual borehole pressure.

Confinement

There is considerable reference to "confinement" in the literature (39,40). The position of the hole relative to the free face and the size and shape of that face can significantly influence the amount of damage; the argument being that the more difficult the breaking the greater the damage. But it is not strictly correct to directly relate confinement to vibration. Peak vibration levels are reached in only a few milliseconds whilst the rock might not begin to move for 20 ms or more. The extra damage due to confinement is probably due to the high gas pressures being more contained therefore extending the inelastic disturbance further. This is one of the most difficult factors to assess as there is very little factual data available although there are frequent references to the detrimental affects of over-confinement.

Conversely the potential to do significant damage to the rock mass is dependent on whether that rock mass has a "space" into which to dilate. If the rock is completely confined and has nowhere to move to, Figure 29, then it is unlikely that structural damage can penetrate very far. But if there is a "free face" then considerable disturbance can occur.

Experience suggests that the amount of significant damage that can result from a particular type of explosive in a certain number of holes could vary four fold depending on the pattern, timing and geometry of the blast (39).

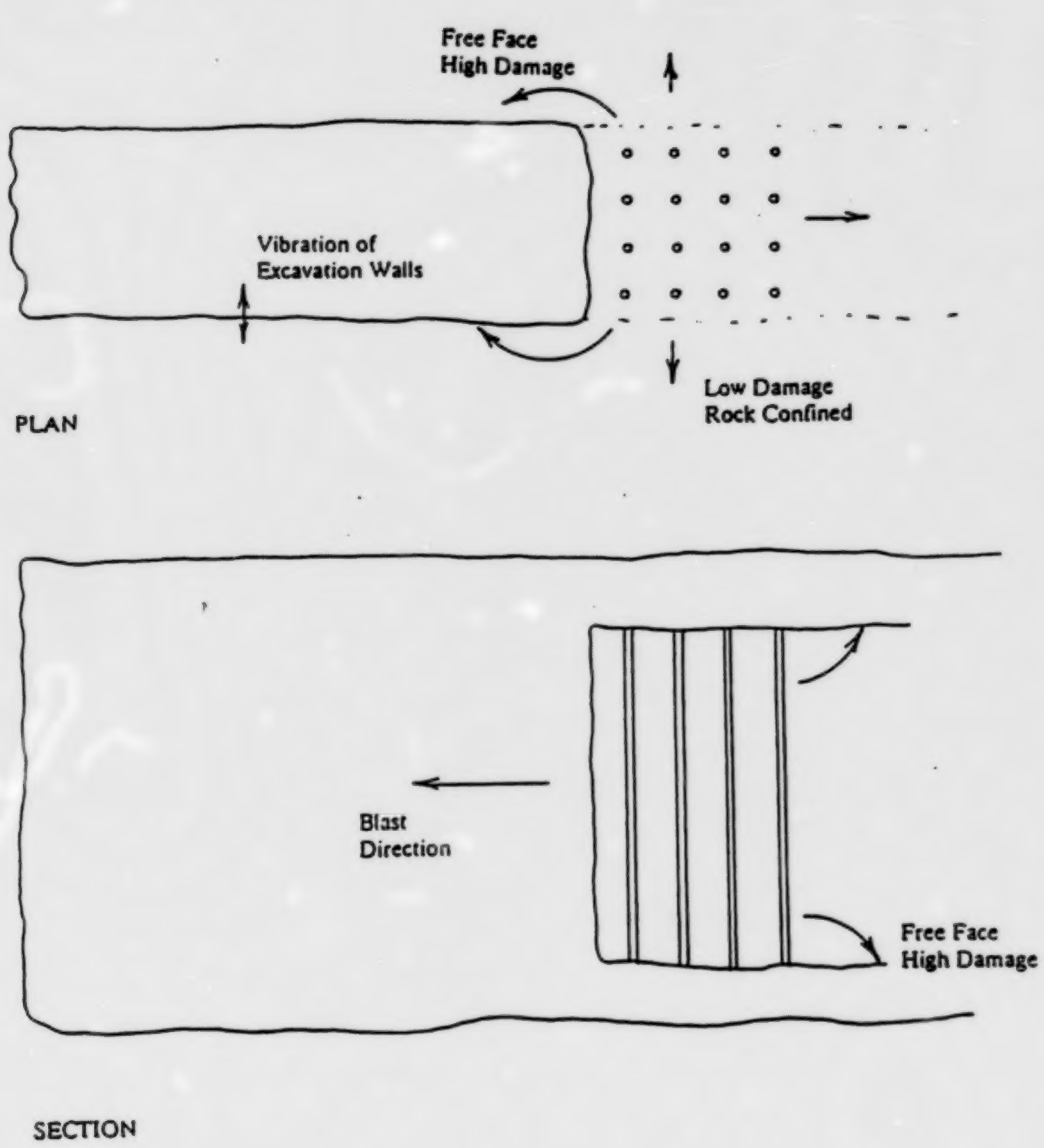

Free Face
High Damage
Vibration of
Excavation Walls
Low Damage
Rock Confined
PLAN
Blast
Direction
Free Face
High Damage
SECTION

FIGURE. 29. POTENTIAL FOR DAMAGE IN DIFFERENT PARTS OF A STOPING EXCAVATION DEPENDING ON FREEDOM TO DISPLACE ROCK.

Timing and Charging

Out of sequence shots, sympathetics and misfires will all affect the level of damage that results from a blast. Misfires and out of sequence blasts will change the confinement and might increase the burden to the extent that a blow-out results. If holes or decks are too close together then sympathetics can occur which will result in a considerable increase in the charge weight which detonates on a single delay.

The delay between detonators will affect the vibration levels dependent on the drill hole pattern and the distance from the measuring point (41). This can become an extremely complicated subject but is of more importance for the protection of areas relatively far from a detonation, of the order of tens of hole lengths. Rock mass damage in open stoping is generally limited to distances less than a single hole length from the blast. In this case the importance of the delay between holes is limited to its influence on confinement. Holes in a successive row should not detonate before the holes in the current row have started moving and provided burden relief.

Many of the problems which occur are due to faulty practice such as having charged holes too close together, damage to downlines or to detonators and insufficient stemming between decks. A further problem in multi-hole and multi-deck blasting is the delay scatter which can result in out-of-sequence firing.

An estimate of the potential damage from a specific blast could be too low by at least a factor of two should misfires and sympathetics occur.

The above discussion concerning explosive power, coupling, confinement, timing and charging demonstrates that a better estimate of the potential for damage can be made than considering charge weight and distance alone.

The next step is to consider whether this potential will actually result in damage.

4.3 Damage Criteria

The primary input for the evaluation of excavation stability is rock mass quality. A number of classification methods are used to rate the quality and these methods include rock mass properties such as:

- intact rock strength
- RQD (rock quality designation)
- joint sets and joint spacing
- joint condition; joint roughness, infilling, water etc.

Blast damage cannot affect the intact rock strength. But it will increase the number of cracks, which in turn reduces RQD, increase the number of sets and frequency of the joints, and decrease the joint shear strength through movement and opening (39).

Different rock masses will have varying resistances to disturbance by blasting and many different thresholds are quoted in terms of peak particle velocity:

- Hoek and Brown (42)

breakage of rock	2500 mm/s
onset of damage	650 mm/s
rock falls in unlined tunnels	300 mm/s
minor damage, cracking of plaster	130 mm/s

- Oriard (34)

no influence on rock slopes	<100 mm/s
occasional falls of loose rock from slopes	130-380 mm/s
damage in weak rock masses	600 mm/s
strong rock may not show damage up to	2500 mm/s

- Holmberg (36)

significant damage to excavation walls may start	600-1000 mm/s

- Isaac and Bubb (43)

acceptable damage in slate 2.0 m from wall	400 mm/s
acceptable damage in slate 0.4 m from wall	1200 mm/s

The potential for damage will also depend, to a large extent, on whether the rock mass has freedom to dilate (39) as illustrated in figure 29. If there is a space into which the rock mass can be moved then considerable damage can occur. A typical example is the backbreak which occurs around the collars of holes in open pit blasting compared with the overbreak at the toe of the holes. The toe is very seldom overbroken although there is a considerable increase in the number of cracks but they seldom extend very far. The rock is very confined at the toe, therefore little or no movement on, or opening across, joints can occur. There are no published values which could be applied to different geometries with regards to damage prevention.

In nearly all the references on rock damage the criteria for applying a constraint to the blasting is by limiting the peak particle velocity. But the limits are very seldom, if ever, related to the rock mass strength or to the stability of the excavation. There are some references to the depth of the disturbed zone (43) (44) (33) but the methods for assessing the disturbance are different in each case and there is no quantitative method for evaluating the impact of the disturbance on the stability of the excavation.

Conservative vibration limits can be accepted in civil engineering but are quite impractical in mining. Any constraint which results in additional excavation costs must be justified by a saving in less support, less dilution or the use of larger stopes. Damage to the rock mass is an extremely important subject but one for which there are no published evaluation procedures which are acceptable to underground mining.

4.4 Conclusions

The major contributor to damage would appear to be the gas pressure. It is difficult to measure the actual pressure at different distances from a borehole and the peak particle velocity is accepted as the best estimate.

Conventional methods of estimating damage by calculating the peak particle velocity would appear to be far too conservative for open stope blasting. Methods must be included which will take into account the explosive strength, the manner of charging and timing and the geometry of the free face relative to the charge. This will give better estimates of the potential to do damage.

The energy constraints usually applied would be impractical in underground mining. The rock mass quality and the estimated stability of the excavation should be taken into account before applying any constraint. Also the importance of the excavation should be considered by including the impact of dilution. Three metres of slough in a wide stope has far less effect than in a narrow stope.

Finally, it will be necessary to calibrate the energy estimates against the difference between the designed stability and the measured stability. This should give an approximate value of the potential to do damage in different rock masses.

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APPENDIX B

ANNUAL REPORT 87/88

B1 TO B22	ANNUAL REPORT 87/88 TEXT
B23 TO B32	FIGURES NO. 1 TO 10

CANADA - MANITOBA
MINERAL DEVELOPMENT AGREEMENT

RESEARCH AND DEVELOPMENT IN MANITOBA MINES

Work on the project was conducted under the auspices of the Canada Centre for Mineral and Energy Technology, Energy Mines and Resources Canada under the Canada/Manitoba Mineral Development Agreement.

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CANMET Scientific Authority:	J. Pathak

TASK 5 - OPTIMIZATION OF BLAST DESIGN FOR FOR BLASTHOLE MINING

Annual Progress Report - 1987/88

by

INCO Ltd., Thompson, Manitoba

July 19, 1988

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1.0 INTRODUCTION

The fiscal year 1987/1988 was the second year of the Task 5 project and includes the time from April 1987 to March 1988. The work completed on the project during this time was mainly involved with the following:

* Literature review update; research continued throughout the year to maintain an up-to-date base of technical information.

* Data collection; the thrust of this work centered around the fragmentation model including inherent fragmentation and the actual muckpile fragmentation.

* Model development; the main portion of development included the selection of algorithms for the fragmentation model.

* Model calibration; the data gathered for the fragmentation system was compared with the predicted results from the model.

* Computer system development; initial developments were produced with a Fortran 77 compiler and compatible graphics software.

The ultimate goals of the project will be reviewed once again to serve as an overview of the Task 5 agreement.

2.0 PROJECT SCOPE AND DIRECTION

The major emphasis of the project will be on developing a rational blast design tool to be used by mining engineers. The approach accepted should yield a user friendly software package with interactive graphics to assist the designer. Important considerations include rock mass properties, fragmentation predictions, blast design, and overbreak. Once the designer is satisfied with the results a method of reporting will be available to display the drill-hole layout and charging details.

It should be noted that a period of calibration will be needed on any mine site due to the large variation in rock mass properties and inherent stress and the difficulty in accurately modelling such inherently variable characteristics. Therefore, the design tool will allow for calibration factors so that, in the end, it will produce reliable results and enable alternatives to be studied.

3.0 LITERATURE REVIEW UPDATE

The majority of the work carried out in this area centered around the fragmentation model, especially in the theories on inherent fragmentation. As mentioned in the formal literature review for Task 5 in 1986, a significant amount of research has been completed by Hudson & Priest. In general terms, their work has indicated that discontinuity spacings in rock can be represented by a negative exponential distribution. The discussion that follows will go into further detail than previous reports have to describe the basics of their model.

Initial work carried out by Priest (1) showed that unless there is a large predominance of evenly spaced discontinuities, a combination of evenly spaced, clustered and randomly positioned discontinuities leads to a negative exponential form of frequency versus spacing value curve. In mathematical terms the form of the negative exponential distribution as related to spacing is:

$$f(x) = L * e^{(-L*x)}$$

where f(x) is equal to the frequency of a discontinuity spacing "x", and "L" is the average number of discontinuities per metre. The above relationship is particularly applicable in geologic zones that have been subjected to a number of complex and different physical events.

In blasting, discontinuity characteristics control the method by which the rock mass will behave. Research carried out by Piteau (2) showed a number of characteristics important in engineering design, and as proposed, in blasting processes. These included:

* The nature of their occurrence (ie. sets or isolated)
* Orientation and position in space
* Continuity
* Intensity
* Surface geometry
* Nature and thickness of the joint fill

One of the few methods that exist for determining each of the items listed above is the scanline survey. Priest (1) indicates that to estimate the number of discontinuities per metre, the scanline length should be at least fifty times the mean discontinuity spacing. Finally, the work showed that at least two hundred measurements were generally required for a clear normal distribution curve.

Hudson (3) has found that the negative exponential distribution characterizes spacings related to random points on a scanline, indicating that the discontinuity positions are mutually independent. Thus, a form of a negative exponential curve will occur because of the tendency for large spacing values to be separated randomly. Likewise, the aforementioned research shows that the relationship is well suited to a mix of discontinuity spacings.

In blasting the resulting fragmentation is very much dependant on the inherent distribution of microfissures, joints, bedding planes, fissures and faults. Priest (4) showed that, in addition to the frequency of the above distributions of discontinuities, a major consideration should be given to the extent of a known discontinuity, or sets of discontinuities. The extent could play a major role in controlling the way in which a rock mass behaves in the blasting process. Given a specific blast design, various results of fragmentation would be found as the extent of the fractures varied from impersistent, non-continuous to persistent and continuous conditions.

The above discussion dealt mainly with the inherent fragmentation distribution and the methods used to determine the relationship between discontinuity spacing and frequency. It is proposed that the distribution will enhance the prediction of oversize in blasted rock. The model that previously gave the resulting fragmentation from blocks of ground that are considered homogeneous is the Kuz-Ram model, as first described by Cunningham (5). Since that time the model has been updated in a number of areas, including rock quality and explosives strength. The revision by Cunningham (6) is given for the rock factor "A" as: $A = 0.06 * (RMD + JF + RDI + HF)$;

where A = rock factor

RMD = rock mass description	powdery/friable	= 10
	vertically jointed	= JF
	massive	= 50

JF = vertical joint spacing(JPS) + joint plane angle(JPA)

JPS	0.1m	= 10
	0.1m to MS	= 20 ; where MS is oversize (metres)
	MS to DP	= 50 ; where DP is drill pattern (m)

JPA	dip out of face	= 20
	strike perp. to face	= 30
	dip into face	= 40

RDI = density influence

RDI = 25 * RD - 50 ; where RD = density (t/m^3)

HF = hardness factor

if Y < 50 GPa, HF = Y/3

if Y > 50 GPa, HF = UCS(MPa)/5

Further modifications were also proposed for the uniformity coefficient "n" used in the Rosin-Ramler equation. Cunningham (6) adjusted this parameter for bench height, different explosives in the hole, and the spacing to burden ratio. The new equation for "n" is presented as:

n = (2.2 - 14B/D) * (((1 + S/B) / 2)^0.5) * (1 - W/B)
  * ((ABS(BCL - CCL) / L + 0.1)^0.1) * (L/H)

where D = hole diameter

B = burden

S = spacing

ABS = absolute value of

BCL = bottom charge length (explosive #1)

CCL = column charge length (explosive #2)

L = total charge length

W = standard deviation of drilling accuracy

H = bench height

In summary it should be noted that the science of blasting is constantly changing and evolving in various ways. Therefore, it is important that the literature review portion of Task 5 continue on until the end of the project.

4.0 DATA COLLECTION

The majority of work carried out in terms of data collection centered around the fragmentation model. Given the high variability of rock mass characteristics from one zone to the next, it was mandatory to develop a good database in this model. Essentially, there are two main components of the fragmentation model. These include the inherent rock mass fragmentation and the resulting blasted fragmentation. Other areas of data collection included detonator scatter time and blast analysis using vibration monitoring.

4.1 Inherent Fragmentation Data

As previously mentioned in the literature review the proposed model for inherent fragmentation is that described by Hudson and Priest (various), based on the research that shows that the frequency of joints related to spacing forms a negative exponential distribution. With this in mind, scanline surveys were completed to gather information on the discontinuity frequency. A sample of the data collection sheet is shown in figure 1.

Once the data was collected it was transferred to a stereonet from which the three most consistent joint sets were determined. The frequency of each set was added up and divided by the scanline length to determine the apparent number of joints per metre. These results were then geometrically corrected for scanline orientation bias to determine the true mean spacing for each joint set. Next, a negative exponential distribution was fitted to the data for each set using the mean spacing and various discontinuity spacing intervals. A Monte-Carlo simulation was then used to generate random sample spacings for each set. The three random spacings, one from each set were then multiplied to give block volumes. Finally, these blocks were analyzed to produce a percent passing curve which will be further discussed in section 5.1 .

4.2 Muckpile Fragmentation Data

Data collection for the actual muckpile fragmentation was completed on a form showed in figure 2. The size ranges selected are a reflection of typical ranges found in a muckpile. These could be adjusted in any direction to suit specific operations or concerns. Numerous data sheets are collected for any given workplace and then tabulated to give an average percent found in each range. These results are then plotted on a percent passing curve.

Other information collected includes a measure of the number of joint sets that can be distinguished on blocks that have faces defined by joints. This will give an indication of the ability of the explosive to create fresh fractures and/or influence the discontinuities in the rock mass. Finally, an estimate is made on the amount of cubic, tabular and elongate material present in the muckpile. Information in this form will be used to compare the shape of a typical fragmented block with that of the inherent rock mass block.

4.3 Blast Analysis Data

A study of blasting practice at Thompson Mine, funded by INCO, was completed and a portion of the data can be used in Task 5. Information was gathered for various blasts using vibration monitoring equipment. Geophones were installed in strategic locations nearby and within the blast to collect the vibrations produced from the blast. The typical layout used to locate the geophones is shown in figure 3. Detailed analysis gave results on peak particle velocity, charge and energy effectiveness, misfire rate, and scatter time on delay initiators.

Summarizing, the results of the study that may be useful to Task 5 are as follows.

* Misfires occurred in all blasts

* Misfires were caused by overbreak, overcharging, in-situ structure, and drill-hole deviation

* Blast execution is complicated by complex decking and sequencing

* Examination of the fragmentation revealed predominately fine material with a number of large boulders. Bi-modal muck size distribution is characteristic of overcharging.

* Blast vibration monitoring can provide an effective means of assessing

blast design and quantifying blast performance.

5.0 MODEL DEVELOPMENT

The majority of work completed in this area involved the development of the fragmentation model. This model enables a prediction of muck size from a blasting process to be made. The two most important items that lead to this prediction are the inherent fragmentation in the rock mass and a well known fragmentation model. The basis for the inherent fragmentation model has been discussed while the fragmentation model is the Kuz-Ram model as described by Cunningham and mentioned earlier.

5.1 Inherent Fragmentation Model

An assessment of the inherent fragmentation can be determined from the data collected on the scanline survey sheet as discussed in section 4.1 . The resulting spacing for each set is used as input for the negative exponential distribution and yields a curve of frequency versus discontinuity spacing as shown in figure 4.

A distribution is produced for each of the three joint sets and then multiplied to give a resulting block volume. Thus, if a spacing curve was sampled 1000 times, a series of block volumes could be produced and represented on a cumulative percent passing curve as shown in figure 5.

5.2 Kuz-Ram Fragmentation Model

The main thrust of the Kuz-Ram model is to provide an estimate of the blasted fragments in various size ranges. A series of four equations are used to develop the results and are described below.

$$R = e^{-(X/X_c)^n} \tag{1}$$

where "R" is equal to the proportion of material retained on the screen,"X" equals the screen size,"Xc" equals the characteristic size, and "n" is equal to the index of uniformity. This equation produces the Rosin-Ramler curve. The parameter known as"Xc" is determined from the above equation based upon 50% passing the mean size range. This is given as:

$$X_c = \bar{X}/(0.693)^{(1/n)} \tag{2}$$

The parameter "$\bar{X}$" is known as the mean fragment size. A relationship was developed by Kuznetsov (7) and is described as:

$$\bar{X} = A * (V_o/Q)^{0.8} * Q^{0.167} \tag{3}$$

This equation is based upon TNT based explosives and has since been modified to ANFO based products. This relationship exists as:

$$\bar{X} = A * (V_o/Q_e)^{0.8} * Q_e^{0.167} * (E/115)^{(-19/30)} \tag{4}$$

where ... X = mean fragment size

A = rock factor

Vo = rock volume

Qe = mass of explosive per blasthole

E = relative weight strength of the explosive

The final parameter in the Rosin-Ramler equation is the index uniformity "n" . It was previously mentioned in the literature review update and contains numerous blasting variables. The index determines the shape of the curve and commonly varies between 0.8 and 2.2 (Cunningham 1983). A more uniform size distribution results from high values of "n" , while a bimodal distribution can occur from lower values. General improvements to "n" and thus fragmentation can be made by decreasing the burden to hole diameter ratio, increasing the spacing to burden ratio, increasing the drilling accuracy, and increasing the charge length to bench height ratio. An example calculation now follows.

Spacing = 3m

Burden = 3m

Drill-hole Diameter = 165mm

Drilling Accuracy = 0.3m

Bottom Charge Length = 0m

Column Charge Length = 14.0m

Total Charge Length = 14.0m

Bench Height = 14.0m

Rock Factor = 10

Volume = 153.0m^3

Explosive(ANFO) = 251.0kg

Rel. Weight Strength = 100

therefore;

$$n = (2.2-(14(3)/165)) * ((1+3/3)/2)^{0.5} * (1-(0.3/3)) *((ABS(0-14.0)/14.0)+0.1)^{0.1} * (14.0/17.0)$$

$$n = 1.95 * 1.0 * 0.9 * 1.01 * 0.82$$

$n = 1.45$

and $\bar{X} = 10 * (153/251)^{0.8} * (251)^{0.167} * (100/115)^{(-19/30)}$

$\bar{X} = 10 * 0.67 * 2.52 * 1.09$

$\bar{X} = 18.4$ cm

also; $Xc = 18.4/(0.693)^{(1/1.45)}$

$Xc = 23.7$ cm

finally; $R = e^{(-X/23.7)^{1.45}}$

thus for:

X(cm)	0.1	0.2	0.4	0.8	1.6	3.2	6.4	12.8	25.6	51.2	102.4	204.8
R(%)	100	100	100	99	98	95	86	66	33	5	0	0

A plot of the resulting cumulative percent passing curve is shown in figure 6.

The proposed fragmentation model for Task 5 is based upon a combination the two previous models. The volume of rock in the Kuz-Ram model is assumed to be homogeneous over a region defined by spacing * burden. However, the inherent fragmentation has created homogeneous blocks of various sizes. An assumption is made that only those blocks intersected by a drill hole will be further reduced in size. The probability that a borehole with a given spacing and burden will intersect an inherent block can approximate what volume of the total volume will be fragmented using the Kuz-Ram relationship. Factors to allow for some crack propagation across joints and to impact will be included once the results of calibration are known. A table is presented which displays the above assumptions (see figure 7). A plot of the resulting fragmentation curve can be seen in figure 8 along with the associated Kuz-Ram and inherent fragmentation curves.

5.3 Time-delayed Initiation Model

A major component of blast design is the timing of the blasting pattern. Errors in the firing times of blastholes can virtually destroy any other well planned blast design parameters, and the final result.

One of the factors that leads to poor timing of initiation is scatter time in the millisecond-delay initiators. Data such as displayed in figure 9 shows that there can be significant differences in the firing time of any one period of blasting cap. Sometimes the differences between two adjacent periods are enough to create overlap causing the blastholes to fire out of order.

The proposed model to determine the probability of overlap is developed from research done by Rholl(8) and the USBM. Their work shows that a reasonable estimation of overlap can be determined by sampling a population of blasting caps, and the resulting distributions can be compared to evaluate the probability of overlap.

A result of their work was a computer program that uses the mean and standard deviation of two blasting caps to yield the probability of overlap. An option in the program enables a crowding factor to be input so that the probability of a minimum interval between firing times is maintained. This program will be used as a base to establish a multi-hole comparison for Task 5.

6.0 MODEL CALIBRATION

An important phase of the project will be to calibrate the blast design model. Inherent variations in the workplace such as rock quality, stress levels, overall geometry, and type of blasting method will undoubtedly lead to significant differences in the results. At this point in time only large variations in results and conditions will be analyzed. Data availability is much to inaccurate to distinguish anything but gross differences, and the quality of data can be variable depending on outside factors (ie. incentive systems, etc.) .

Calibration will include analysis of data sheets and photographs and will concentrate on areas with potential for large variation. These areas are summarized below.

1. Wide Stope) similar blasting method
   Narrow Stope) similar rock mass

2. V.C.R.) similar rock mass
   Slash) similar geometry

3. High Stress) similar blasting method
   Low Stress) similar rock mass, geometry

The results of the calibration process will show a number of variables that tend to be site specific. Finally, methods of determining these "calibration factors" will be outlined to form a basis for site calibration.

7.0 COMPUTER SYSTEM DEVELOPMENT

The majority of work carried out in this area was also centered around the fragmentation model. As well, initial developments were made on a cap scatter program to predict overlap. The coding was all done in Fortran 77 with a compatible graphics software. The objective is to develop each of the models separately and then tie them together, in the end, to form a blast design tool for practical use.

One of the models nearly completed deals with inherent fragmentation and the shape factor associated with it. The program needs input consisting of the mean spacings of three joint sets to model a negative exponential distribution.

A sort is performed on each of the inherent blocks so that the dimensions of the shortest, intermediate and longest sides are determined. Using a diagram developed by Folk (9), a scatter plot is produced based on a triangular relationship of shape factor as shown in figure(10). The three axes represent different functions. They are determined from the following formulas:

Left axis = S/L

Bottom axis = (L-I)/(L-S)

Right axis = (S^2/L*I)^1/3

The plot is useful in giving the viewer a feel of the shape of the inherent blocks. It has been noted by various authors that the shape factor of broken rock is an important item when considering percent passing analyses. The results should be of use in calibration the fragmentation model in that the difference between the inherent shape factor and the blasted shape factor is a measure of the ability of the explosive to penetrate discontinuities.

The fragmentation prediction model as proposed for Task 5 is in the coding phase presently and should be ready for calibration in the near future. The input for the model has been described previously and will yield output in the form of fragmentation curves and other associated numerical values. The associated graphics will be available on a printer and computer screen.

The final portion of the computer system development includes the formulation of a code for cap scatter predictions based on the work completed by the USBM and mentioned in section 5.3. Improvements will be made for graphics output and user-friendliness.

8.0 SUMMARY OF WORK 1988/1989

As this is the last year scheduled for the completion of Task 5 it is intended that the blast design model be finished. This will include the final version of the computer aided blast design program complete with user manual and description. In addition guidelines for safely blasting "hang-ups" in ore/waste systems will be included in the final report.

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SCANLINE SURVEY

LEVEL __________ DATE__________

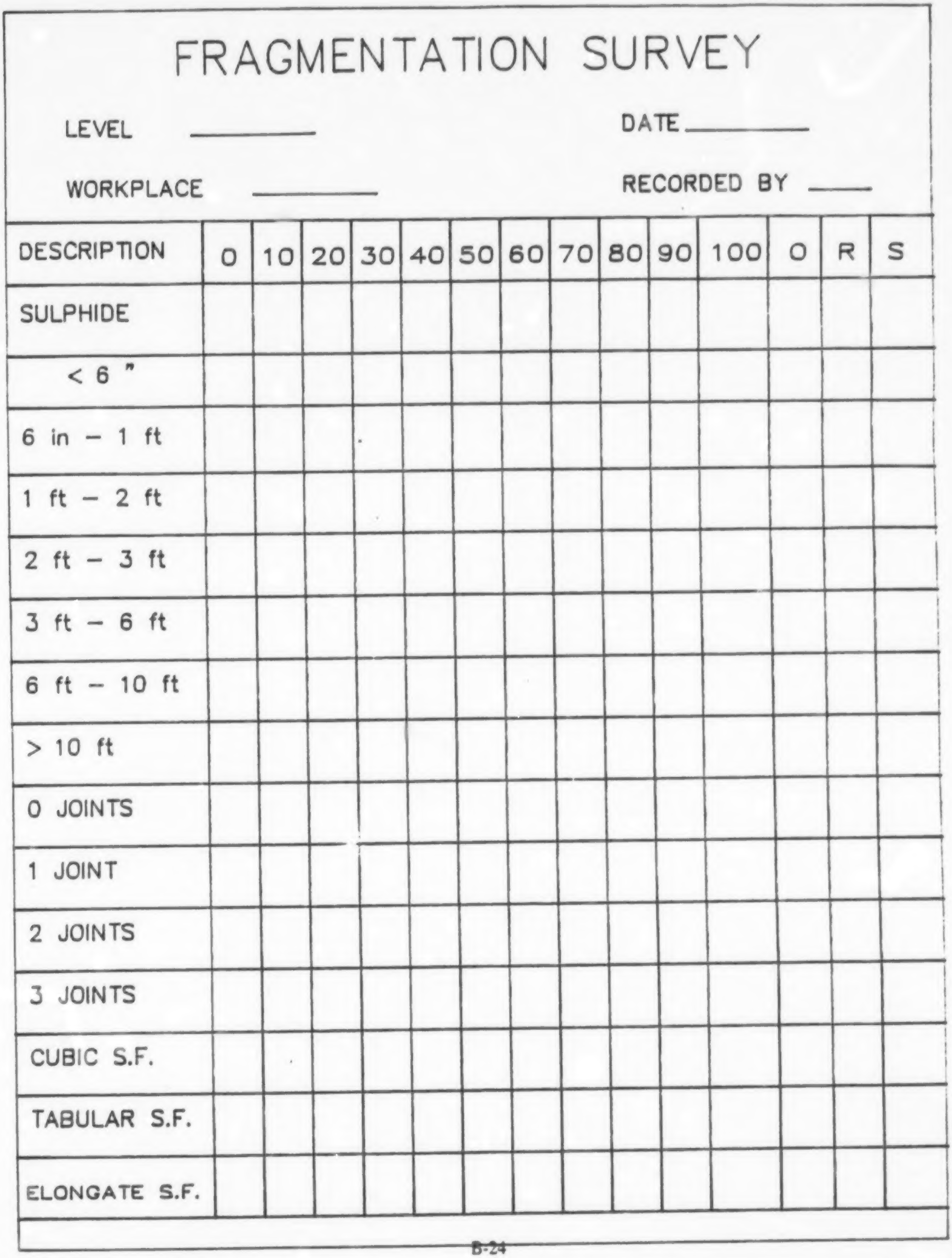
WORKPLACE __________ RECORDED BY _____

LINE LENGTH __________

LINE ORIENTATION __________

SET NO.	DIP	DIP DIR.	FREQUENCY	COMMENTS
1				
2				
3				
4				
5				
6				
7				
8				
9				
10				
11				
12				

FIG. 1



FRAGMENTATION SURVEY

LEVEL ________ DATE ________

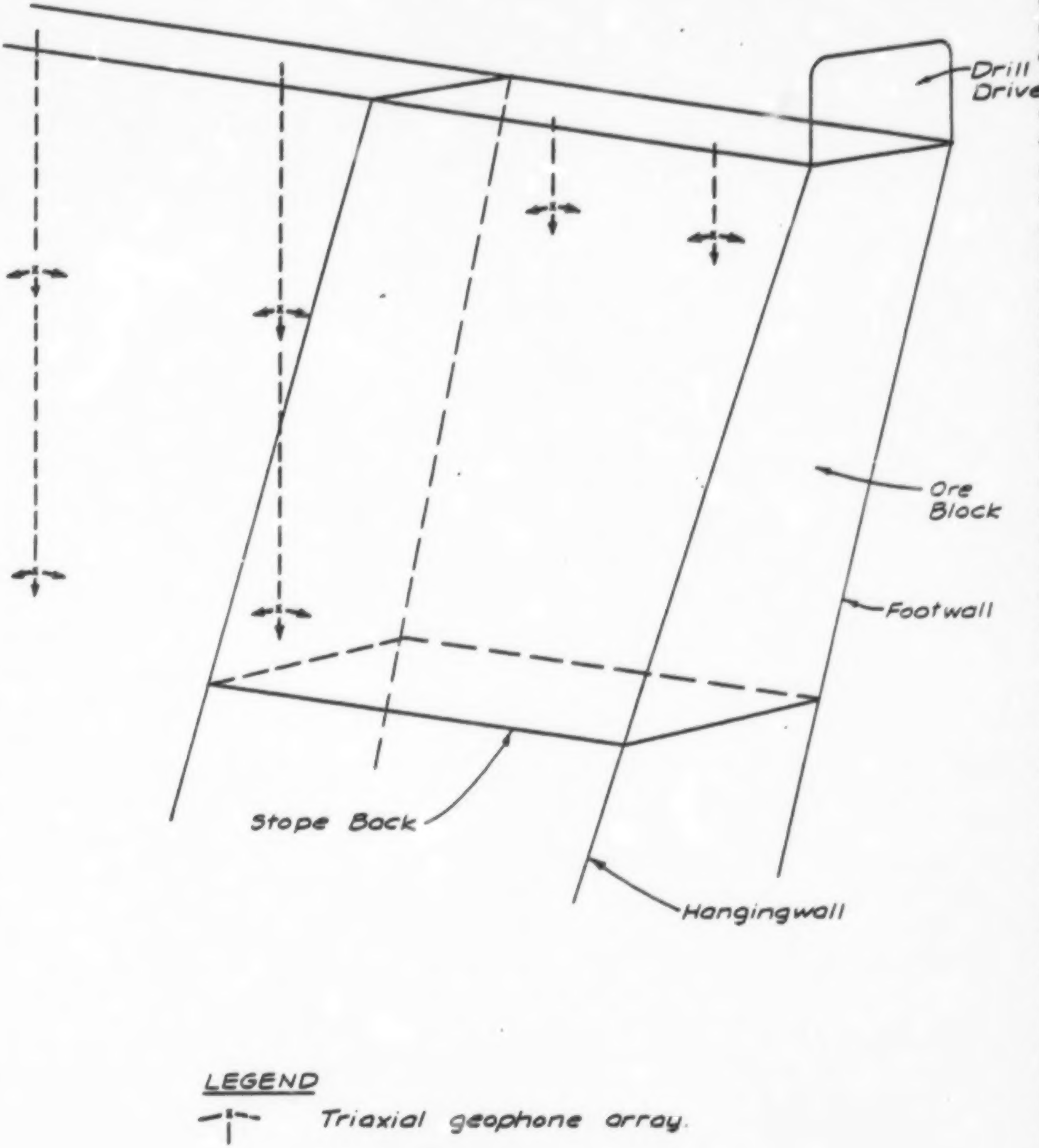
WORKPLACE ________ RECORDED BY ____

DESCRIPTION	0	10	20	30	40	50	60	70	80	90	100	O	R	S
SULPHIDE														
< 6 "														
6 in – 1 ft														
1 ft – 2 ft														
2 ft – 3 ft														
3 ft – 6 ft														
6 ft – 10 ft														
> 10 ft														
0 JOINTS														
1 JOINT														
2 JOINTS														
3 JOINTS														
CUBIC S.F.														
TABULAR S.F.														
ELONGATE S.F.														

FIG. 2

VBM STOPE MONITORING ARRANGEMENT

FIG. 3


Drill Drive
Ore Block
Footwall
Stope Back
Hangingwall
LEGEND
Triaxial geophone array.

PROJ NO. 872-1427 DRAWN R.D. REVIEWED .... DATE Sept. '87

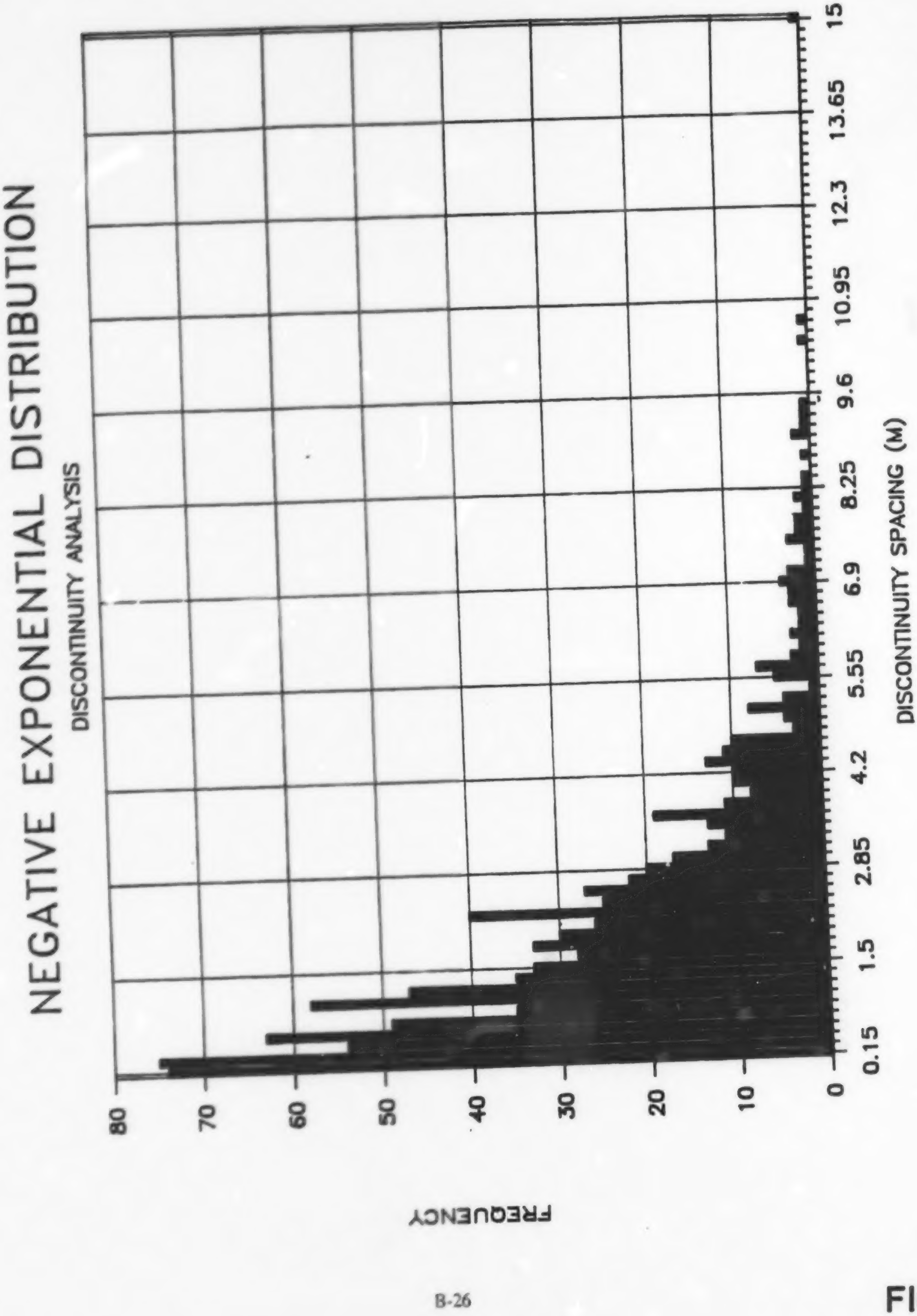

NEGATIVE EXPONENTIAL DISTRIBUTION
DISCONTINUITY ANALYSIS
FREQUENCY
80
70
60
50
40
30
20
10
0
0.15
1.5
2.85
4.2
5.55
6.9
8.25
9.6
10.95
12.3
13.65
15
DISCONTINUITY SPACING (M)

FIG. 4

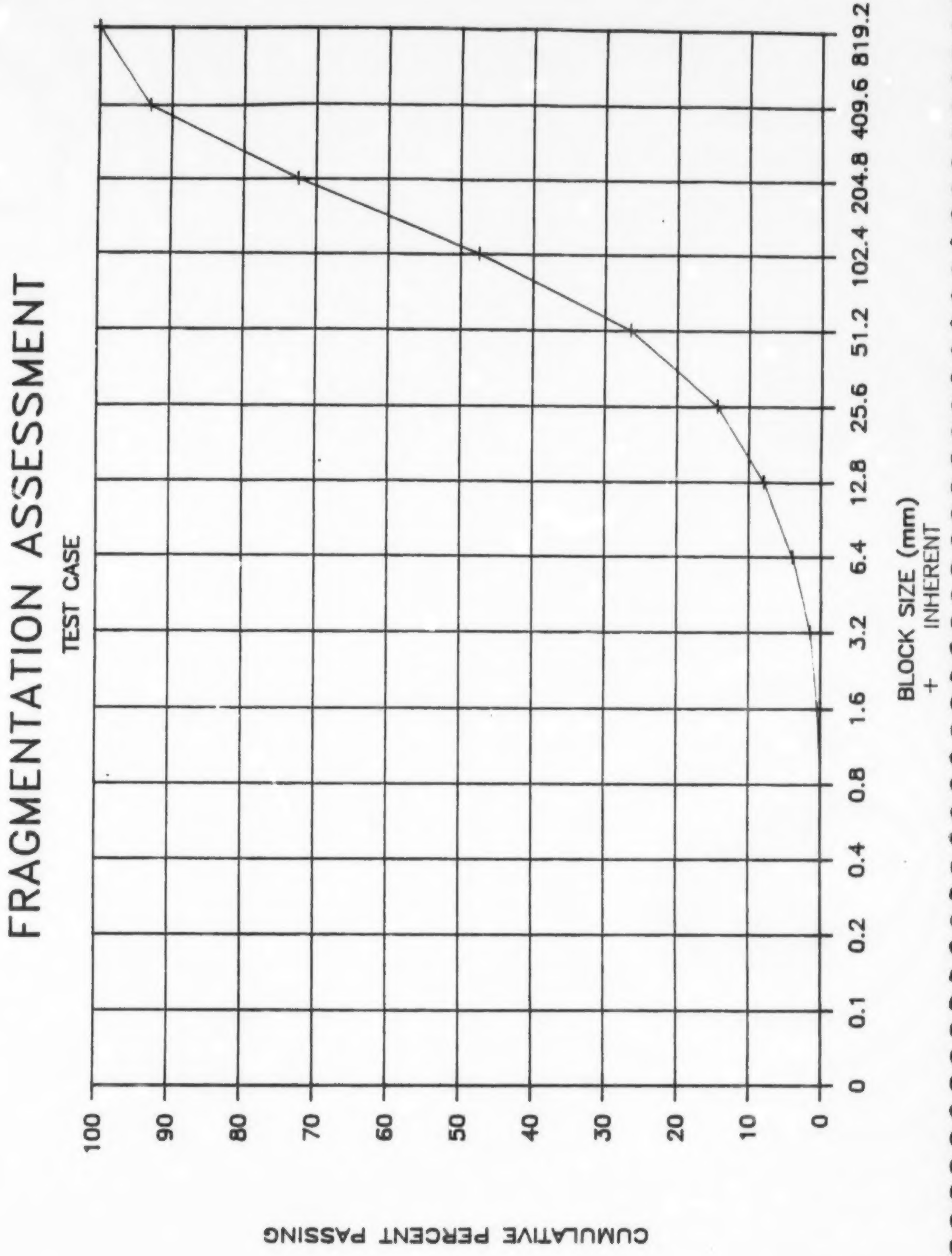

FRAGMENTATION ASSESSMENT
TEST CASE
CUMULATIVE PERCENT PASSING
100
90
80
70
60
50
40
30
20
10
0
0
0.1
0.2
0.4
0.8
1.6
3.2
6.4
12.8
25.6
51.2
102.4
204.8
409.6
819.2
BLOCK SIZE (mm)
+ INHERENT

FIG. 5

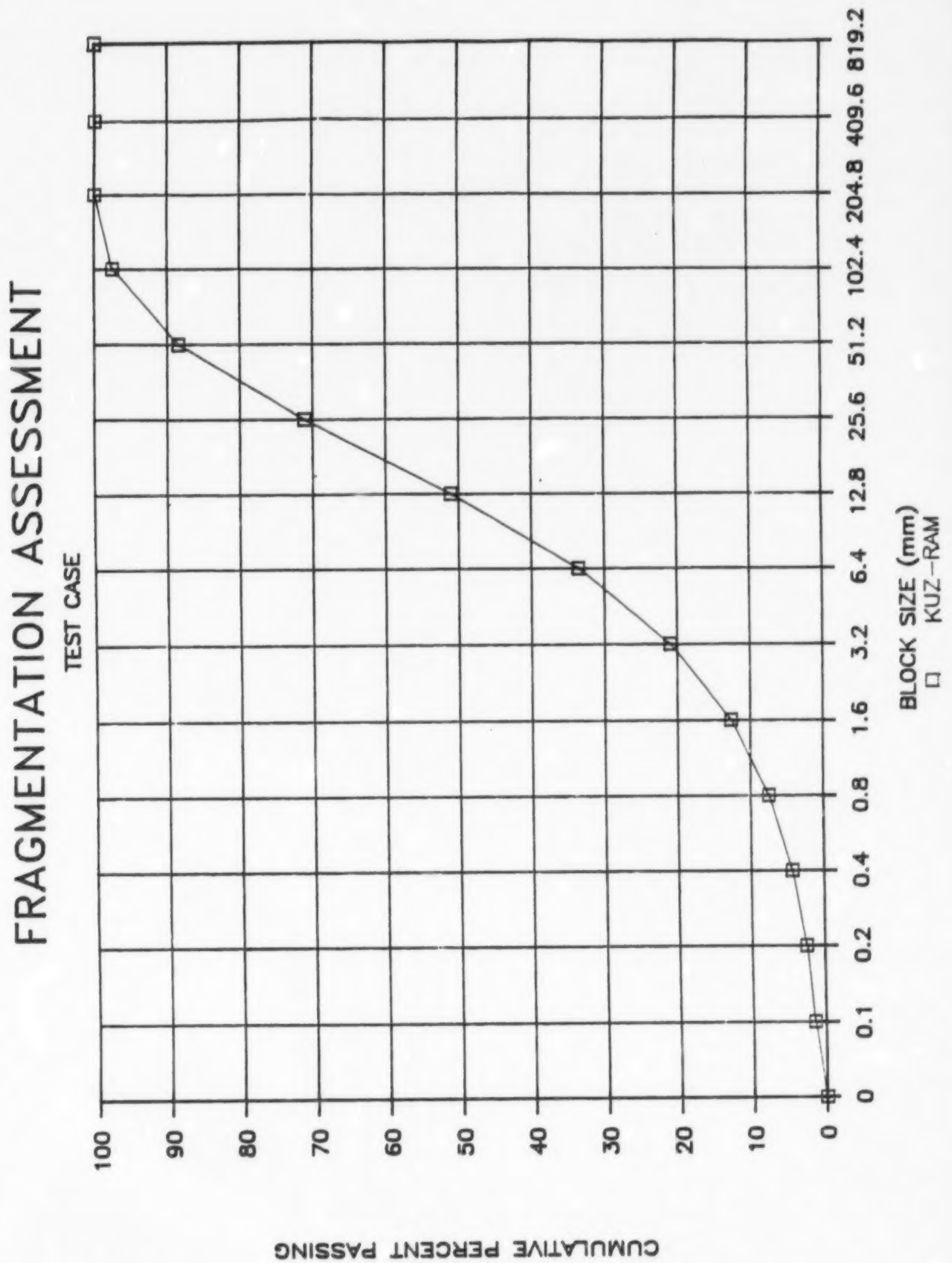

FRAGMENTATION ASSESSMENT
TEST CASE
CUMULATIVE PERCENT PASSING
100
90
80
70
60
50
40
30
20
10
0
0
0.1
0.2
0.4
0.8
1.6
3.2
6.4
12.8
25.6
51.2
102.4
204.8
409.6
819.2
BLOCK SIZE (mm)
□ KUZ–RAM

FIG. 6

TASK 5 FRAGMENTATION MODEL

SIZE RANGE (M) A	KUZ-RAM PASSING (%) B	INHERENT PASSING (%) C	VOLUME 100 CU.M. D	AREA INH. BLOCK E	DRILL PATTERN AREA F	PROB. INTERSECT. G	VOLUME AFFECTED H	VOLUME UNAFFECT. I	KUZ-RAM VOLUME J	TOTAL VOLUME K	TASK 5 FRAG (%) L
0.000 - 0.016	13	0.5	0.5	0.0002	9	0	0	0.5	4.87	5.37	5.5
0.016 - 0.032	8	1.0	1.0	0.0008	9	0	0	1.0	3.31	4.31	9.5
0.032 - 0.064	13	2.5	2.5	0.003	9	0	0	2.5	4.87	7.37	17.0
0.064 - 0.128	17	4.0	4.0	0.013	9	0	0	4.0	6.82	10.82	28.0
0.128 - 0.256	20	6.5	6.5	0.052	9	0.01	0.07	6.43	7.79	14.22	42.0
0.256 - 0.512	17	12.0	12.0	0.210	9	0.02	0.24	11.76	6.82	18.58	60.5
0.512 - 1.024	10	21.0	21.0	0.839	9	0.09	1.89	19.11	3.51	22.62	83.0
1.024 - 2.048	2	25.0	25.0	3.355	9	0.37	9.25	15.75	0.97	16.72	100.00
2.048 - 4.096	0	20.5	20.5	13.422	9	1.49 **	20.5	0			
4.096 - 8.192	0	7.0	7.0	53.687	9	5.97 **	7.0	0			
	100.0	100.0	100.00 M3				38.95 M3	62.05 M3	38.95	100.00	

** PROBABILITY THAT AN AREA > 9 M2 WILL BE INFLUENCED BY A S x B > 9 M. IS 100%

A = LOG SCALE SCREEN SIZE PASSING
B = KUZ-RAM PASSING (%)
C = INHERENT FRAGMENTATION PASSING (%)
D = VOLUME OF INHERENT BLOCK PER 100 M3 (M3)
E = MAXIMUM AREA OF INHERENT BLOCK (M2)
F = SPACING x BURDEN (M2)

G = PROBABILITY OF S x B AFFECTING INHERENT BLOCK (%)
H = VOLUME AFFECTED BY S x B (M3) = G x D
I = VOLUME UNAFFECTED BY S x B (M3) = D - H
J = KUZ-RAM VOLUME (M3) = B x ΣH
K = TOTAL VOLUME PASSING (M3) I + J
L = TASK 5 CUMULATIVE FRAGMENTATION PASSING (%) = $L_x + L_{x-1}$

FIG

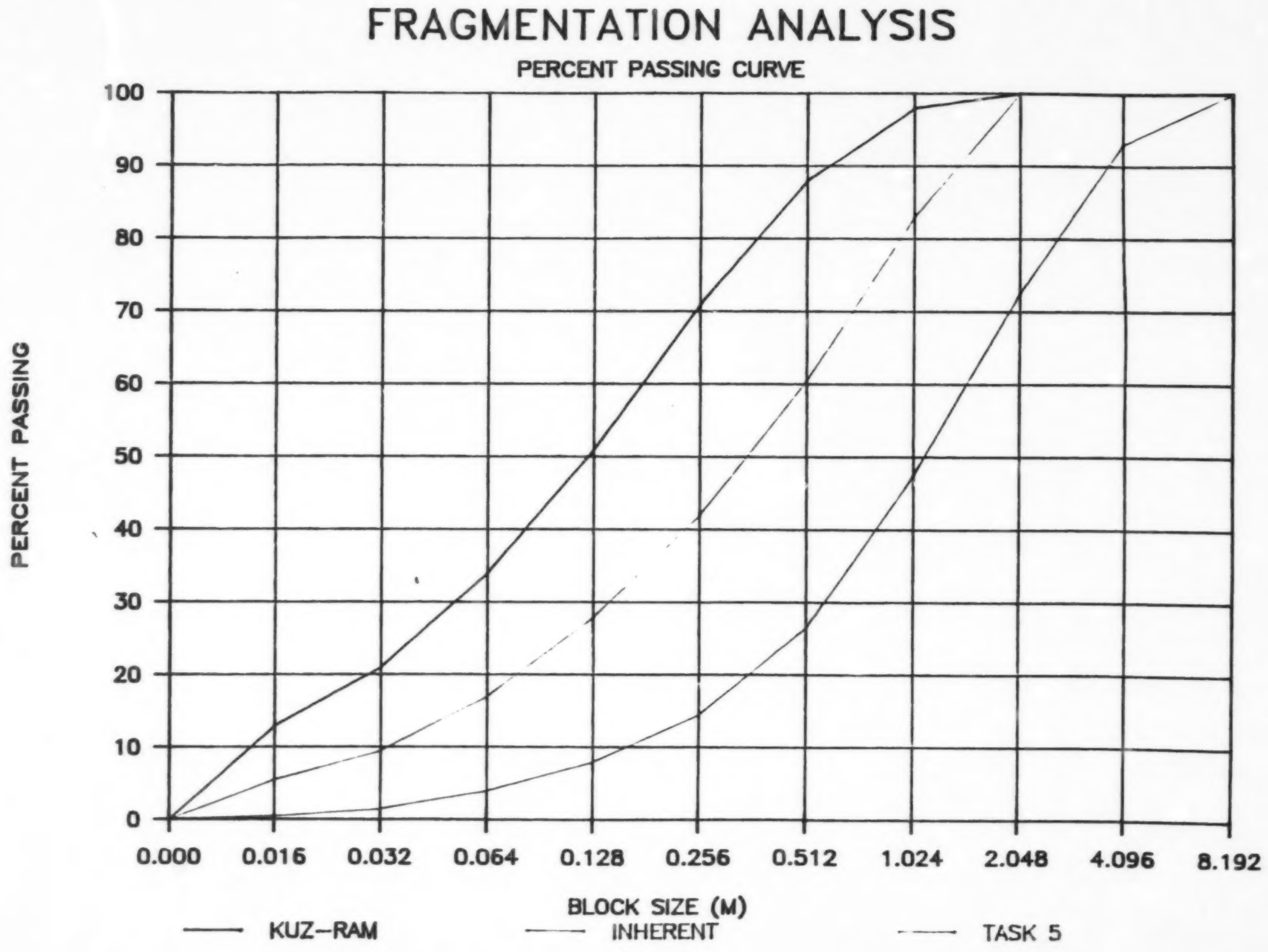

FRAGMENTATION ANALYSIS
PERCENT PASSING CURVE
PERCENT PASSING
100
90
80
70
60
50
40
30
20
10
0
0.000
0.016
0.032
0.064
0.128
0.256
0.512
1.024
2.048
4.096
8.192
BLOCK SIZE (M)
KUZ-RAM
INHERENT
TASK 5

FIG. 8

SUMMARY OF INITIATION TIMES, VBM STOPE BLASTS

Delay	Nonminal Firing Time(ms)	Blast # (ms)	Blast #2 (ms)	Blast #3 (ms)	Blast #4 (ms)	Blast #5 (ms)	Blast #6 (ms)	Blast #7 (ms)	Blast #8 (ms)	Blast #9 (ms)	Blast #10 (ms)	Mean (ms)	Standard Deviation	Mean Error (%)
1	30							36		44	39	39.7	4.0	+32.3
2	50						55	59		63	54, 57	57.4	3.6	+14.8
3	75						76			80	80, 83	79.7	2.6	+6.3
4	100					113	109	102		123	113	112.0	7.5	+12.0
5	128					151	131	129		150	145, 148	142.3	9.7	+11.2
6	157					184	166	167		182		174.7	9.5	+11.3
7	190	251, 258	186	175	210	215		207	215	209	207, 210	213.0	24.0	+12.1
8	230		227		259	246	244	245	243	253	226	243.0	11.4	+5.7
9	280	291, 298	263	269	334	310	269	290	302			288.9	23.3	+3.2
10	340	350	361	374	382	360	350		378	361	341	365.2	16.4	+7.4
11	410	407	418			433	421	416	414	416	417, 449	421.2	12.5	+2.7
12	490	508, 514	517	515	503	520	522		503	518	525, 534	516.1	9.3	+5.3
13	570	602,607, 615	610	585, 610	607	633	579	570	613	616	611	604.5	17.0	+6.1
14	650	710,735	701	699	701	675	697	693	708	662		698.1	19.7	+7.4
15	725		741	776	744	734		753	753	769		752.8	15.2	+3.8
16	800		813,822	819,827	842	833			841	834		828.9	10.4	+3.6
17	875			906	918	920		852	926			904.2	30.2	+3.3
18	950			1043	1024	1027		925	981,989, 1001			998.5	39.3	+5.1
19	1025				1072	1104			1072,1078			1081.6	15.4	+5.5
20	1125				1175,1195	1231		1162	1173,1185			1186.9	24.4	+5.5
21	1225				1317	1272						1294.6	31.7	+5.7
22	1400	1462,1508			1505	1474			1338,1483			1461.7	63.1	+4.4
23	1675	1792			1794				1782			1789.3	6.4	+6.8
24	1950				1995,1999							1997.0	2.8	+2.4
25	2275	2321,2495 ,2546										2454.0	118.0	+7.9

FIG. 9

BLOCK SHAPE DIAGRAM

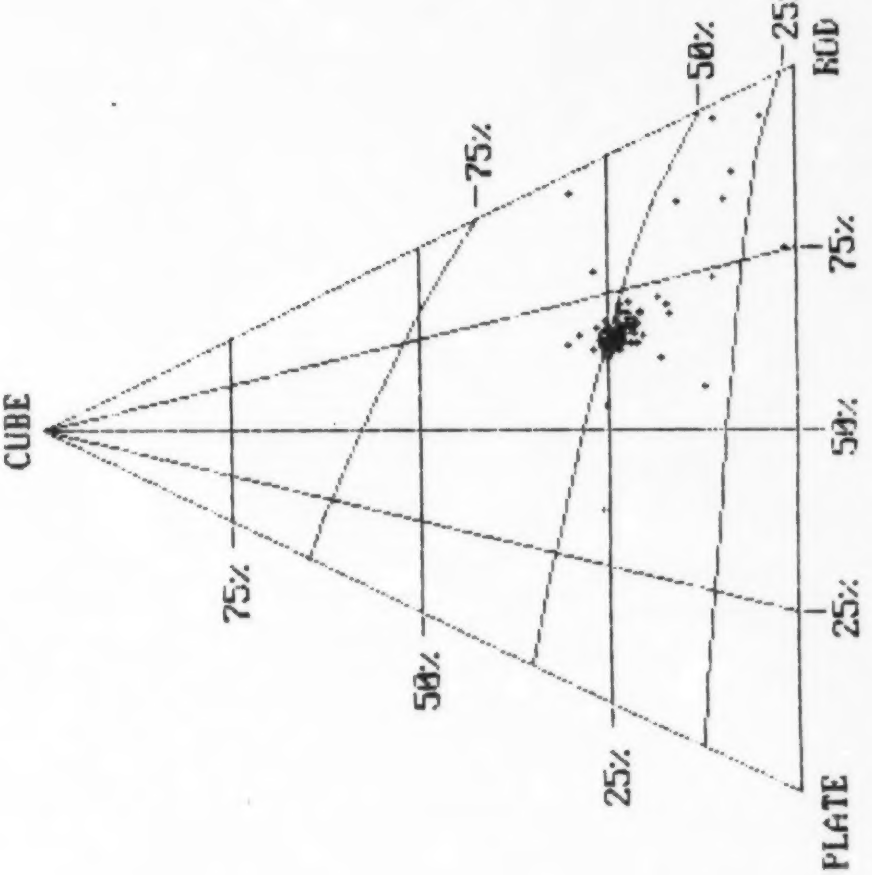


FIG. 10